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Metallurgical & Chemical Engineering

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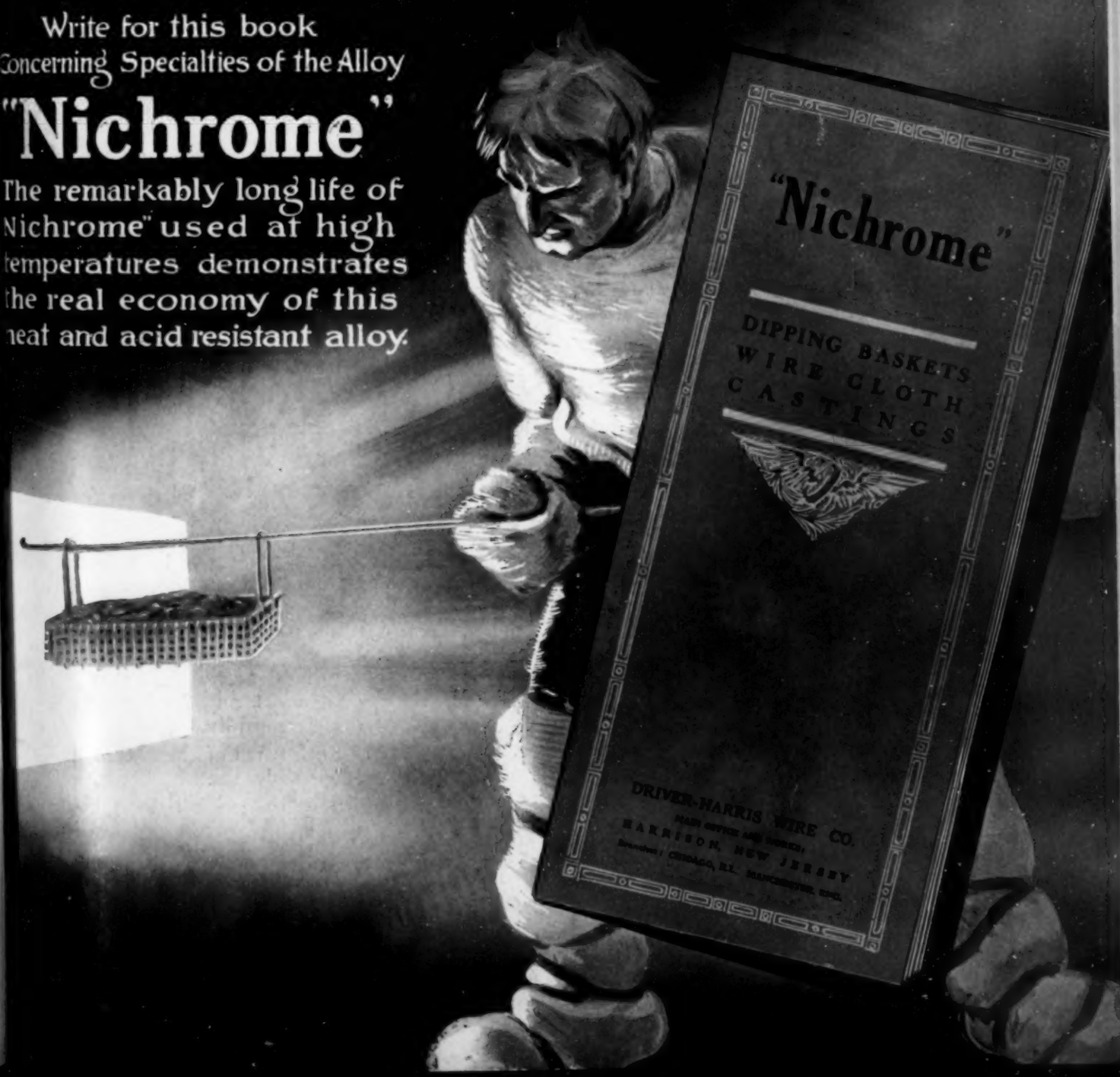

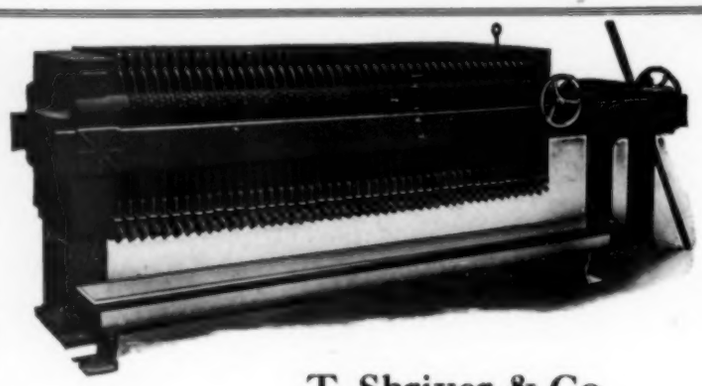

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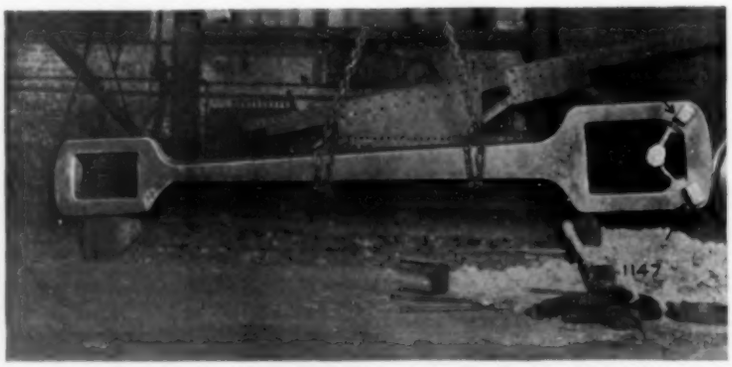

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Second Exposition of Chemical Industries

Le roi est mort, vive le roi. When the exceedingly successful First National Exposition of Chemical Industries passed into eternity a few weeks ago, quick action in preparing the next one became imperative, and we are glad to announce that at a meeting of the Exposition Management and its Advisory Board, held last week, it was definitely settled that the Second Exposition will be held from Monday to Saturday, September 25 to 30, 1916. The management will start at once in its work. But it seems to us that now is the time also for the different national chemical societies, representing the American chemical profession, to start in some preparatory work of their own. The Second National Exposition will undoubtedly be visited by a great many more chemists and engineers from all over the country than have come together anywhere in the United States for a long time. It would seem the logical thing for the national chemical societies to hold, each of them, a meeting—a regular or a special one—at the time of the Exposition in New York. Such a Chemical Convocation Week would be a delightful affair and would emphasize that with all their characteristic individualities which should be faithfully conserved, the different American chemical societies are a harmonious and enthusiastic unit in their ultimate endeavor.

Unprecedented Times in Steel

For the present situation in the steel industry there are no parallels in history. It is easy to make the comparison to test the accuracy of such a statement, for the steel industry as the principal purveyor of a wrought ferrous material is only about a quarter century old. It is scarcely twenty-five years, in other words, since the number of puddling furnaces in operation began to decrease.

Last November and December witnessed greater stagnation in the steel-making industry than had ever before been witnessed. The production of finished steel in December represented less than 35 per cent of the productive capacity. The production of crude steel slightly exceeded the production of finished material, as a few concerns were stocking ingots or billets. The production of pig iron reached a somewhat lower ebb still, as there was considerable stocking of pig.

Thus the market movement in steel is barely eleven months old, but wonderful things have occurred in that period. Since 1907 there have been two major movements in the steel market, in 1909 and 1912, but the present movement has already overtopped them. In the earlier stages of the present movement steel prices advanced at about the same rate as in those movements, but prices have now been advancing for

a longer period and the advance has become more rapid. Finished steel prices, on an average, are now about $2 a ton above the top reached in 1912, and are level with the top reached in 1909, and the curve of prices, instead of showing signs of soon rounding an apex, is trending still more sharply upward. Next we must consider the 1905-6-7 movement. Its top was $4 to $5 a ton above the present price level, but more than two years were required even to approximate that top. In the next preceding price movement, 1901-2, the advances were far from spectacular. In 1899 there occurred a "runaway market" but the spectacular feature was simply the fancy prices that were paid for relatively small quantities of steel for prompt shipment.

An outstanding feature of the present steel market is that as the weeks pass the steel mills are more and more reserved in the matter of booking contracts for forward delivery. Their transactions are being confined more and more to the making of specific sales, the buyer furnishing specifications and all details as to making shipment, which is to occur practically at mill convenience. This change in sales policy paves the way for rapid price advances.

Views as to the fundamental causes of the heavy pressure upon the steel mills have been changing rapidly in the past two or three weeks. When steel demand began to increase, in last December and the first two or three months of the present year, the common thought was that a reaction had naturally occurred, from an unprecedentedly stagnant condition, and there was doubt as to whether the improvement would last long. Later, in the summer, the demand for steel was regarded as very largely a "war demand." The development of the very recent past is that there is a very strong demand from strictly domestic sources for peace purposes, and the steel industry now staggers under its load, for the demand for war purposes is increasing rather than decreasing.

We shall now see tried out on a large scale the experience of a positive demand for steel with no restraint as to prices. For many years the policy of the steel trade had been to view the steel market as something that should be held steady, extreme advances or declines being regarded as highly objectionable. Price advances were likely to curtail consumption and price declines likely to disturb confidence, both therefore inviting disaster. From another viewpoint it might be argued that if the demand for steel is real, the need of each particular consumer should be weighed against the general demand; if it was less in intensity, that buyer should wait until he could buy at a lower figure, and thus there might be great fluctuations in prices, but little fluctuation in production. Such a theory has not found much support in steel selling circles, but a runaway market has been forced upon the industry and through no choice of anyone the experiment is going to be tried. As prices are now advancing, levels will be reached at which some consumers will certainly find it necessary to defer their purchases. Eventually they will have an opportunity to buy at lower prices, and if they take advantage of the opportunity the production of steel will be kept at a steadier rate than has been the case in the past. It may require years for the full period of the experiment but its progress will certainly be watched with intense interest.

A Little Confusion of Thought and Speech

In the profession of mining engineering there are such phrases as "ore in sight," "ore blocked out," "positive ore," "probable ore," "possible ore," to denote the differences between the expected and the actual and to help one to estimate in the mind the degree of certainty of evidence or of proof. The old adage, "There is nothing certain except death and taxes," leads us to believe that there is nothing given man between a reasonable expectation and futuristic determinism. In the chemical and metallurgical professions, where we all know the rapid change that is taking place, there is a similar difference between things as we hope they will be, things as we try to make them, and things as they actually turn out.

Possibly the forceful character of the American nation cannot often discriminate between things as people desire them in the future and things as they really prove to be. If a man repeatedly says, "I will get there" and "This shall happen," he in time may instill into his mind a certain self-confidence now so aptly termed "auto-suggestion." Possibly, indeed very likely, the widespread national looseness about the use of "will" and "shall" is derived from our somewhat egotistical state of mind. In the first person, "will" means "wish" or "desire," "shall" simple futurity, whereas in the second and third person the meanings are reversed and "shall" denotes desire or determination, while "will" means simple futurity. In colloquial speech, not one person in a hundred makes any distinction in the use of "will" and "shall," and the reason is probably that there is no difference noticed in the mind.

Indeed, it is not hard to detect in this fault of speech the fault of looseness of thought and immoderate egotism. Between the intense desire to have the future happen as one wills it and active efforts to make it, there is in one aspect—that of the personal equation—no difference. Fortunately, in the spoken word, the inflection of the voice tells much of the meaning of the speaker. In the written form a certain purity and precision of diction should tell one to be careful, for "nobody shall help me from mistakes of the pen, and I will drown in literary fashion," to paraphrase the classical example of the rhetorician, and in a scientific profession like that of the chemical and metallurgical engineer his language should be as precise as his thoughts.

Saving Trouble in Handling Units

In view of the universal present endeavor to do everything in the shortest possible time and to effect a standardization wherever possible, the psychological moment seems to have come to adopt single systems of units which will be used by everyone. Every engineer knows the extra work which he has had to perform in trans-

ferring from one to another system then back to the first system, and so on, thus increasing the probability of error every time a transfer is made. A good forward step in the right direction in regard to metallurgical practice has been taken by the Mining and Metallurgical Society of America, a preliminary report of whose committee will be found on page 828 of this issue. While everyone may not agree with all of the recommendations made, yet everyone will admit that the single system is the only system for rapid and efficient work and accurate intercourse.

The recommendations to use the metric system of weights and state the gold and silver contents of ores in decimal fractions, or in percentages and decimal fractions, wherever the metric system has been adopted as legal and in all Spanish-American countries, and the recommendation to use degrees Centigrade in a like manner are timely, as everyone who has used these systems knows their immense superiority. The ton, which in this country is so elastic, is recommended to be placed at 2000 lb. The bullion and assay value units should prove a great boon if commonly used. Several units which are in very common use, and have been so for a long time, could be changed in general practice only with greatest difficulty, and this is a great pity. It is to be deplored that the gallon is in such common use that it had to be recommended as a standard, also the "fluid ton," in the case of cyanide solutions, which is stated to closely approximate 2000 lb., instead of exactly equalling 2000 lb.

The recommendations are not intended radically to convert long-used units into units which have not been in general use in metallurgy, since, as before stated, it almost seems practically impossible to change current practice. The idea is to have one system, to adhere to that system, and to endeavor to bring it into general use. There are, however, so many different systems represented in the different units that it is hard to see how many laborious calculations can be avoided in the use of the present recommendations. They are, no doubt, the best for the present, but it is to be hoped that in the course of time we may talk only of liters, kilograms and other metric units.

While You Learn from Mistakes, Make Your Mistakes on a Small Scale

It is an easy and fascinating task to talk about the glories of a research or an invention—how the inventor sees a vision, goes into a trance, and does some quick, hard thinking, and lo and behold, he makes a model and a million dollars. Good brilliant ideas are easy and delightful, but the practicalization of these ideas is, as Rudyard Kipling says, "another story." Actually the popular imagination is far wrong; it sees only the glitter and glamor, not the tedious hard experimenting, not the financial troubles, not the difficulties that beset the path of the research pioneer.

What, indeed, is the keynote of success of a successful invention? And where shall we find the wisdom of the successful one and the understanding that, actuated by a will of iron, persists till the goal of commercial operation is reached? How does the idea become crystallized? There are many answers to this, all correlated and interdependent.

Possibly Edison's classic and laconic answer, "Genius is 2 per cent inspiration and 98 per cent perspiration," is as good a one as can be devised. What we would propose in the psychological atmosphere of the present days is somewhat different, and reads like this: "Make your mistakes on a small scale!"

This is good advice from a business standpoint, for all successful processes are founded on mistakes, and it is necessary to know by actual test where the pitfalls are in order to avoid them. If we assume the wisdom and truth of the slogan "make your mistakes on a small scale," we can adduce as a consequence this: act on your mistakes by the use of your imagination. The human mind is supposed to have several parts, or better expressed, different powers, by which it acts, but for progress or for full delight of life nothing is more a requisite than imagination, tempered by reason and controlled by common sense, for it is that mental ability that enables the possessor to project thoughts like a cinematograph on the screen of the soul. Without this ability to project, to extrapolate a curve far beyond the known, to take the known facts and find out and prove the existence of new facts, no inventor would succeed. And so in scientific transcendentalism do we find the peculiar mental trait that utilizes the past mistakes as stepping-stones to victory.

Now all this operation must be done commercially, for if the invention costs as much to produce as it is worth to society, there should and cannot be on the average any profit in the venture. So the truth of the remark: "Make your mistakes on a small scale."

While a strong motive in building on mistakes may be the personal gain of the interested parties, yet the secondary and often primary motive, if we are to judge by the effects, is the good that society will gain. This must be where the will to believe and the will to conquer has its source.

The personal equation naturally is the driving force. The ability to fight, to sacrifice himself and may be his family, to work with a smile at a thankless task—these are the victories of peace and "the moral equivalent of war." With the imagination apperceiving, there is associated a feeling or intuition, a full development of the feminine kind of thought that tells the inventor how to meet emergencies and how to foresee the future. A successful inventor tells us that at most critical times he acted against the advice of his friends and against his own logical judgment and was right. For in truth can it profit a man to make mistakes on a small scale or on a large scale and not learn from them?

It is in the commercial consideration of the subject that the element of cost must be held in mind. Large mistakes are expensive, small ones are cheap, but the inventor must learn. And he only learns and advances by use of a "will-pervaded" imagination that sees the unknown and attains the impossible. With all this idealistic side, the homely sentence, "make your mistakes on a small scale," blends in harmonious thought.

Readers' Views and Comments

Chemical Engineering as an Independent University Course at Purdue

To the Editor of Metallurgical & Chemical Engineering:

SIR:—I note with interest the editorial comment in your issue of Sept. 1, relative to chemical engineering.

Supplementing the information contained therein, it may not be amiss to state that at this institution, Purdue University, the School of Chemical Engineering has since 1911 existed as a separate school on an equal basis with the other engineering schools of the university, and now numbers sixty-one graduates.

The course is attracting a superior class of men, and in spite of a very strict interpretation of entrance requirements, the present enrollment is 168.

The curriculum of the School of Chemical Engineering may be found on page 84 of the catalog of the university. It may be said in regard to the work of the junior and senior years that these are now in a period of transition which will result in physical chemistry being administered as a required subject, beginning with next year.

H. C. PEFFER.

School of Chemical Engineering,
Purdue University, Lafayette, Indiana.

Tube-Milling: Closed Circuit

To the Editor of Metallurgical & Chemical Engineering:

SIR:—The discussion on tube-milling in your February number is interesting. Closed circuit would seem to be with us to stay, at least for quite a while; so were fine feeds and high discharges, but after a while they went their way.

There is no doubt that the economical result obtained should guide, and as shown in the discussion of the article by Mr. Haynes and his table of results, a cumulative load of 570 tons will give a net grinding to minus 200 of 58.59 tons. And the relative time that each ore particle is in the tube-mill may be fourteen minutes, that is, for each round; but does it not circulate six or seven times and in the end be eighty-four to 100 minutes in the tube-mill anyway? Is there any advantage in circulating pulp so much if the same result can be had without so doing? It costs something to return pulp to the tube-mill, even when using drag classifiers. Is there not something radically wrong with any system of crushing when it is necessary to return the pulp so many times in order to reach a desired result?

As there can be no argument over the crushing machine used; i.e., that the tube-mill, with its load of pebbles, furnishes all the elements necessary to successfully break or grind sand of six or finer mesh down to minus 200, we must look into the condition of the feed in order to find some condition that can be changed so that better results may be had.

In the earlier years of tube-mill experience it was generally thought that the feed—the battery discharge—had to be relatively fine in order that the tube-mill might be able to do its work. Thirty-mesh screen on the batteries was quite common. Nowadays most mills use coarse screen, six, eight, ten or twelve-mesh on the batteries. This change in general custom may be considered proof that tube-mills can successfully grind more net tons of coarse feed than they can of fine feed. It is in order then to find the reason why this should be so.

May it be that between two pebbles equal volumes of coarse pulp and of fine pulp will be caught, and in each case the total volume be reduced to ultimate size; and that the volume of coarse pulp weighing more than the same volume of fine pulp, there is a greater resulting tonnage ground in the case of the coarse pulp? I believe that this is what takes place, and while total volumes are not really ground at once to ultimate size yet equal proportions of such volumes will be ground, so the same relative result is obtained; more coarse pulp is reduced in the operation than would have been the case had the pulp been fine. This, of course, implies coarse feed, reasonable in size, such as the six-mesh and nearby sizes, when compared with the sizing of the pulp returned from the mill when fed with overload.

I cannot agree with any theory that a six-mesh particle of ore must go through a tube-mill five to seven times, being somewhat reduced on each passage, until it will finally emerge a 200-mesh particle. I think that if it once lodges between two pebbles in action it will at once be ground or broken so that the larger portion of it will be ground to minus 200, and the remainder will be approaching that size and must again be caught between other pebbles in order to be finished to 200-mesh. Poor grinding occurs because ore particles go through the mill without being caught between grinding surfaces and not because the force applied is insufficient to reduce the particles at one operation; there is an excess of force applied, but much of it is not utilized, as only those particles of ore actually between the rubbing surfaces are fully ground, and those particles that lodge just outside the points of contact get only partial grinding.

With pulps of mixed sizes of ore particles, the coarse sand will come into play or be exposed to grinding conditions before the accompanying fine sands. The coarse sand parts with its water more readily and clings better to the pebbles; its comparative diameter also is greater and there is less chance of its passing between the pebbles; a fine particle might escape entirely where a large one would be caught, and being caught it is at once reduced greatly to a finer mesh. In the succeeding part of the mill all ore particles are approximately of the same size, most of the fine feed going through without any grinding, and to that is added the fine stuff left from breaking down the coarse part of the feed, so we go on accumulating return load until the gross load fed to the mill is something out of all reason.

When we reduce an ore sample in a mortar with a pestle we reach a point in the operation where we can no longer apply force of impact and get any results: we must resort to force of abrasion. When we put the pulverized sample on the bucking board we do not strike the pulp with the muller, we rub it or rather grind it by abrasion. It is universal custom to reduce sands and ore pulps for assay purposes to sixty, eighty or 100-mesh by abrasion, yet in tube-milling many seem to think that to reduce such pulps to 200-mesh force by impact should be applied. And so they contend that the mill revolutions should be such that the pebbles fall clear of the upcoming load. With the body of the mill half full of fluid pulp there cannot be much force by impact expended when the falling pebbles reach bottom after passing through the thick pulp floating in the mill. With pulps as dilute as 38 to 40 per cent moisture most of the fine parts of the ore will be in suspension, due to the violent agitation, and the coarse sands will have settled to the bottom where they are ground by abrasion, not impact.

Grinding pans are in many places considered superior

to tube-mills for producing minus 200-mesh pulp; they crush by abrasive force, not by impact. If we wish to obtain abrasive grinding in tube-mills the revolutions must be so adjusted that the pebbles just begin to cascade, but the main body slides back over itself without falling clear.

Battery sand is of relatively small size when compared with the irregularities or projections on the surfaces of tube-mill pebbles. Sand caught between two pebbles struck together is more likely to escape crushing than to receive it; if the same two pebbles were rubbed together the sand would more likely be crushed than to escape, as the inequalities on the two pebble surfaces would more likely fit into one another than to rest on the points while in motion.

As to sufficiency of force applied, I think that the weight of the load of pebbles, coupled with the dragging exerted by the liners causing the pebbles to push one against another, is ample to crush ordinary six-mesh and finer sizes. If crushing of ¼-in. and larger sizes be attempted force by impact may be necessary.

In order to secure the fullest results in grinding, the character of the pulp should be such that no segregation of parts can occur; coarse sands, fine sands and the water should remain evenly mixed, then all sizes get an equal chance of entering the grinding zones. With dilute pulps the coarse sands separate and settle to the bottom, where they come into close contact with the pebbles. The fine sands tend to be held in suspension, float up in the main body of the mill, and get only casual contact with the pebbles. With pulps of around 30 per cent moisture no segregation of parts takes place—the whole pulp clings to the surface of the pebbles, is carried along with them throughout their travel, and gets every opportunity for being caught in the grinding zones. There will be less floating load in the body of the mill as most of the pulp will be traveling with the pebbles. It will probably cause an unbalanced load and require more power to turn the mill, but there will be no cushioning of the grinding forces if "cascading" be practiced—every pebble works every moment. I think that with cascading of the pebbles 90 per cent of the grinding done is accomplished by abrasive force exerted between the pebbles in the bottom of mill and on the uphill movement. With 38 per cent moisture no clinging of the pulp to the pebbles can occur; sands settle out quickly and are ground while in the bottom of the mill, the fine product then rising to float in suspension with the main body of the pulp.

Scoop-feeding solved all the troubles incidental to charging thick pulps into tube-mills; low discharges solve those incidental to passing thick pulps through tube-mills. Thirty per cent moisture pulps will not flow freely in launders, but screw conveyors will pass them. Thirty per cent pulp comes out of the tube-mill as a thick ooze, lining the chamber 2 in. thick, but this can be diluted by a hose playing on the outside of the cylindrical screen. We found that 25 per cent moisture pulp would work through a mill having 18-in. discharge end, but could not maintain an even flow with as much tonnage as the mill would fully grind; we could maintain the flow for hours at a time, long enough to determine to our satisfaction that such low moisture would be desirable if we could overcome the mechanical difficulties of handling it. We thought that with lower discharge we could have obtained the tonnage desired, and with open circuit. With 30 per cent moisture we could maintain an even flow of pulp, but the tonnage desired could not be reached, though we did obtain better than 80 per cent through 200 mesh.

According to the experiments of W. B. Easton, with a mill equipped with lifter devices he obtained 70 per cent through 200 when passing 144 tons through it with open circuit, and with 240 tons gross, closed circuit, he obtained 72 per cent through 200 mesh. These are figures that I believe have never been equalled in tube-mill practice. I would take it that the moisture contents were 38 per cent and 40 per cent, and would like to have seen his results when passing pulp of low moisture. His experiments would indicate that, with low discharge, sufficient feed of any moisture can be passed through a mill in quantity to overload, and that consequently the closed circuit cannot be dispensed with entirely; but when net grinding to minus 200 reaches such figures as 70-odd per cent it would seem that it would be better to work one or perhaps two tube-mills, following in series to take the concentrated return from all the primary mills.

The figures given by Mr. Easton concerning minus 200-mesh product are all in conjunction with 38 per cent and 40 per cent moisture pulps; we would like to have had some of his figures or results of passing low-moisture pulps. His comments on low moistures would infer that he favored them for their grinding effects; he declares himself in favor of them as regards their requiring lower power consumption, and he also states that fuller loads develop lower power consumption. I am inclined to think that the effect of the lifters will be to extract the top of the floating load in the mill first, and thereby lower the moisture content of the charge of pulp in the mill at a given moment. I consider the lifter device the greatest improvement in tube-mill design that we have seen in a long time.

Guadalajara, Mexico. W. J. PENTLAND.

The Naval Consulting Board

To the Editor of Metallurgical & Chemical Engineering:

Sir:—I have read with interest the current literature and news in regard to the Naval Consulting Board and it has occurred to me that the question might well be raised in connection with this whole matter as to why the Consulting Board should devote its attention only to naval matters, as apparently intended, rather than to both Navy and Army needs. The step taken is in the right direction and will undoubtedly produce splendid results, will promote patriotism and promote in general a personal interest in the affairs of our country such as has been lacking to a large extent in the recent past. In general it seems that our government has been left too much to the politicians.

Without doubt the wonderful success of the Germans at the present time is due to the fact that they make the most of the technical skill and scientific brains available in the German Empire. In our own country Army and Navy matters have been left largely to inexperienced men and those who may be in more or less of a rut in the government service. The best men of the country have been employed in industrial pursuits almost entirely and have not apparently devoted much attention to our national protection, and national protection should came first in as much as our industrial success is dependent upon this.

Why should not the Consulting Board's Navy Research Laboratory be a joint Navy and Army undertaking? It would seem that there could be only one answer to such a question.

L. F. VOGT.
Standard Chemical Co.,
Canonsburg, Pa.

Mineral production in California in 1914.—The complete detailed report of the mineral yield of California for 1914 compiled under the direction of Fletcher Hamilton, State mineralogist, is now ready for distribution in the form of Bulletin 70 of the State Mining Bureau.

Benzol and Toluol from California Oils

The Standard Oil Company of California is producing now on a small commercial scale the much cherished benzol and toluol.

Strictest secrecy is maintained as to methods applied in producing from the California crude oil, which has an asphaltum base, these materials so much needed in the dye industry, as well as in the explosive manufacture.

Apparently there is a very handsome profit in the manufacture, and there is some talk of the oil company eventually entering into the dye industry.

Another new product the Standard Oil Company of California has to its credit is a product sold under the trade name of *"calol,"* a perfectly water-white, tasteless and stable oil used for medicinal purposes.

The famous British surgeon, Sir Arbethnot Lane, who discovered "kinks" in the intestines of human beings, found that in milder cases it was not necessary to operate, but that the "kinks" could be overcome if a suitable internal lubricant could be supplied. "Russian oil" was the remedy, but since the war this supply has been cut off, and many inferior substitutes have been placed on the market. After lengthy experiments an oil has been produced by this company which is equally good and has even been pronounced better than the Russian product.

The demand for the product is great, and while the large plant is being built a small temporary plant is operating to its capacity, not beginning to be able to fill orders.

The world's demand of this internal lubricant is being supplied at present from California and from California crude oil.

Nitrogen Fixation in Russia

It is reported in Scandinavian papers that a large company with strong capital backing has been formed in Russia for the purpose of manufacturing nitrates from the nitrogen of the air.

The process to be used is the well-known Birkeland and Eyde process.

All arrangements with the Norwegian company, owners of the Birkeland and Eyde patents, have been made and a plant is being built in the North of European Russia, off the White Sea.

It is interesting to note Russia's entering the electrochemical field in this manner, and also that through dire necessity of war an industry is started that in time of peace produces products that are a blessing to the inhabitants of its country, and in war times produces the same products, but they are then no blessing, but rather a curse to the inhabitants of adjoining countries, "Les extreme ce touche."

The Willard Gibbs Professorship of Research in Pure Chemistry

In course of the academic year 1914-1915, a new department was established on a permanent basis in the Mellon Institute of Industrial Research of the University of Pittsburgh, namely, a Department of Research in Pure Chemistry. The headship of this department is to be known as the Willard Gibbs Professorship of Research in Pure Chemistry, "to ever proclaim the ideal which the incumbents of the chair and the groups of research workers to be associated with them will be expected to follow." It constitutes a chair in the Graduate School of the University of Pittsburgh as well as in the Mellon Institute.

Dr. Martin A. Rosanoff, who had built up and for years headed the Graduate Department of Chemistry in Clark University, was formally inaugurated as the first permanent incumbent of this professorship, at exercises held in the Assembly Hall of the Mellon Institute on Oct. 26. The exercises included addresses by Prof. James McKeen Cattell and Prof. Marston Taylor Bogert, both of Columbia University, and by Dr. Rosanoff, followed by a reception and an inspection of the Mellon Institute.

Chancellor Samuel Black McCormick presided at the exercises. He explained the circumstances connected with the establishment of the chair and thanked the donors whose generosity had made possible the endowment, of which the income is to constitute the salary of the incumbent of the chair. He then introduced Dr. Cattell.

Standardization of Metallurgical Units

A preliminary report of the Committee on Standardization of the Mining and Metallurgical Society of America contains suggestions pertaining both to mining and metallurgical matters. We give below those recommendations relating particularly to metallurgy.

Mesh of Wire Cloth.—The standard mesh of the Mining and Metallurgical Society of America shall be the standard screen of the W. S. Tyler Company of Cleveland, Ohio, as described in that company's catalog No. 36. The basic screen of this scale has an opening of 0.0029 in., which is assumed to be equal to 200 mesh. The opening in each successive screen increases in the ratio of the square root of 2, or 1.414.

This furnishes a standard screen scale in which the area of each successive opening in the scale is doubled that of the second preceding.

The dimensions of the Tyler screens are as follows:

Openings		
Millimeters	Inch	Mesh
26.67	1.050	
18.85	0.742	
13.33	.525	
9.423	.371	
6.680	.263	3
4.699	.185	4
3.327	.131	6
2.362	.093	8
1.651	.065	10
1.168	.046	14
0.833	.0328	20
.589	.0232	28
.417	.0164	35
.295	.0116	48
.208	.0082	65
.147	.0058	100
.104	.0041	150
.074	.0029	200

Weights and Measures.—1. For all Spanish-American countries, and all other countries where the metric system of weights is the legal one, a statement concerning measures and weights of whatsoever kind shall be in the metric system.

All references to temperature shall be in degrees Centigrade.

In Spanish-American countries and in all other countries which have adopted as their legal standard of weight and measures the metric system, the gold and silver contents of ores, alloys, bullion or other materials, must be stated in decimal fractions of the weight of the material, or in percentages and decimal fractions thereof or else in kilos of silver and grammes of gold per metric ton of material.

2. The word "ton" shall represent a weight of 2000 lb. avoirdupois (29,166.6 oz. troy).

It is advisable to abandon the use of such terms as "hundredweights" and "quarters," and to express fractions of a ton in pounds or decimals of a ton.

3. The term "miner's inch" shall represent a flow of 1.5 cu. ft. of water per minute. Where this unit is not customary and currently acknowledged, the flow of water shall be expressed in cubic feet per minute or per second.

4. In the United States the unit of liquid measure is the standard gallon of 231 cu. in. In England, and the British colonies which have legalized the unit, the "Imperial gallon" measure of 10 lb. of water is to be used.

In every case all reports should state definitely which gallon is used therein.

5. Temperature should be expressed in degrees Centigrade. Subsidiary use of the Fahrenheit scale is admissible; but only conjointly with the Centigrade scale and not alone.

6. Returns of gold and silver shall be expressed in terms of fine gold and silver respectively, not as "bullion."

7. Gold and silver contents of ores determined by assay may be expressed in money values per ton of 2000 lb. avoirdupois, as well as in weight per ton. In this connection the value of gold shall be taken at 85s., sterling, or $20.67 U. S. currency, per troy ounce of fine gold.

All other constituents of alloys, metallurgical products, or other material must be given in percentages of the substances analyzed.

It is to be noted that certain commodities, such as pig iron, steel rails or large sections, etc., are dealt in in various arbitrary units. Where it is a recognized custom of the trade to use such special unit, statements therein are permissible, but provided, however, that it is always clearly stated what the unit be in some other and standard unit.

Bullion and Assay Values.—The following is proposed concerning these terms:

1. All samples should be dried at 100 deg. C. before being submitted to assay or analysis and reports should definitely state that this precaution has been taken.

2. Assay values of gold and silver ore and products should be represented in ounces and decimals and not in ounces, pennyweights and grains. They shall always be expressed in terms of fine gold and silver respectively and not as "bullion."

3. Assay values of alluvials shall be reported in grains and decimals of a grain of fine gold, or in pence (at 2d. per grain of fine gold), or in cents per cubic yard.

It is recommended that, in the absence of specific information as to boulders, etc., 1 cu. yd. of ordinary alluvial be taken as equivalent to 3000 lb. (1.5 short tons).

4. In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution.

In the case of cyanide solutions, the use of the "fluid ton of 32 cu. ft." is recommended. It closely approximates to 2000 lb. and is in common use.

5. When it is necessary to state or estimate the money value of an ore, etc. (other than gold), the contents shall be given in percentages of the base metals, and in ounces of gold and ounces of silver.

If it becomes necessary to estimate the value of the ore in money the first step would be a deduction from these percentages to allow for metallurgical losses and the next following step the calculation of the value of the recoverable metals at definite fixed prices. If the net value of the ore is required then from the above should be deducted the working and realization costs.

6. We recommend for all laboratory sieving the Tyler standard screens.

The Aniline Dye Situation

In the account of Mr. I. F. Stone's paper before the New York Section of the American Chemical Society in our issue of Oct. 15 (page 710), the statement on the increase of aniline dyes manufactured in this country was incorrectly reported. We quote the section in question verbatim from Mr. Stone's complete paper:

"Another large increase in production through the present conditions is the manufacture of aniline colors and other coal-tar products in this country, which will be much to the relief of consumers, who at the present time are unable to get anything like the supply of colors which they need in the conduct of their business. The five factories already established in this country before the war are all extending their production to their utmost capacity in keeping with safety for their investment, and I believe that in 1916 the production of aniline colors in this country will be at least three to four times the production before the war.

"This production could again be largely increased if the manufacturers were sure of some protection from the Government in the way of higher tariff or Government manufacture of intermediate materials, which would enable them to get the same at the same price as paid by European manufacturers, and then again, the putting into effect of some anti-dumping clause, which is now promised by the Government, to prevent the dumping into this country of colors at lower prices than they are sold elsewhere for the purpose of preventing their development and manufacture here.

"Whatever the Government may finally decide to do toward the protection of this industry, there remains the fact that a great impetus in the development is already under way, with the hope of Government protection, so that the Europeans will find them strongly entrenched in any event after the close of the war. I am speaking now of the factories already in operation, but in addition to these there are many new factories projected and some in operation. I have a list of twenty-three new factories, the last of them with a proposed capital of $15,000,000, and while a good many of these may not materialize, something will surely come of some of them.

"Up to now none have actually manufactured any aniline colors, but some are operating with a production of some intermediate products like aniline oil, betanaphthol, paranitranilin, etc., and speaking of aniline oil, with the one factory established before the war and the increase in its works since the war, and the number of new factories making or about to make aniline oil, it looks as if the production of this product would be 8000 to 10,000 tons, as against the normal consumption of about 4000 tons; in other words, the proposed manufacture seems larger than the consumption, but as the consumption is also increased possibly the whole amount projected can be used. At any rate, there will be enough manufactured in this country to take care of the whole consumption, so that Europe need not be depended upon."

* * *

With respect to the report of Dr. THOMAS H. NORTON'S paper before the New York Section of the Society of Chemical Industry (page 780 of our issue of Nov. 1) it should be stated that Dr. Norton made no direct recommendation of a Government factory for the production of coal-tar intermediates. What he did say in reference to Mr. Stone's plan in this respect was as follows: "The various admirable features of the plan, including the well-organized preparedness for the speedy manufacture of explosives and munitions generally, in case the nation be menaced by invasion, render the project worthy of the most careful study."

As to the proposed league of American consumers of dyestuffs, Dr. Norton did not recommend that all consumers who would purchase imported dyes at dumping prices should be penalized or ruled out altogether. He said: "It can also be easily seen that a powerful weapon is put into the hands of those who collectively represent the consumers of colors, to rigidly rule out from the

sources of supply any manufacturer, foreign or domestic, who would deliberately make an attempt to undersell his American rivals, or who would resort to any method of unfair restriction of trade."

American Iron and Steel Institute Meeting at Cleveland

The ninth general meeting of the American Iron and Steel Institute held in Cleveland, Oct. 22 and 23, was a success in every particular. The meeting was a record breaker in attendance, about 680 having registered during the two days. The reduced number of papers was a pleasing feature and helped to sustain the interest and discussion. On Friday evening a banquet was held at the Hotel Statler, at which moving pictures were shown of manufacturing processes at the plants of the National Tube Company and the American Steel and Wire Company. After this feature, Judge Gary called on a number of prominent members for short addresses. Saturday morning several hundred of the members enjoyed a visit to the plant of the White Company, and a special train was provided to take the members to the Pennsylvania ore dock, where the unloading of the "J. P. Morgan" of the Pittsburgh Steamship Company, was witnessed.

The first session was opened Friday morning with an address by Judge Elbert H. Gary, in which he laid special emphasis on publicity, on the harm done to large corporations through their failure to take the public into their confidence. "The general public and private individuals have been in the past too far apart for the good of both. Lack of knowledge often breeds unnecessary and hurtful antagonisms, and many have suffered even though they were not personally at fault." . . . "There should be published whatever of business matters the public is legally entitled to know and also whatever may affect the public interest and may be exposed without detriment to the corporation or individual concerned." Judge Gary said we should have protection against the results of cheap labor and impoverished conditions abroad, in order to enjoy continued prosperity after the war.

Mr. Samuel Mather, chairman of the local committee, was the first speaker called on by Judge Gary. Mr. Mather spoke of the history of Cleveland and the development of the steel industry. Cleveland ranks fifth among the cities of the country in the value of manufactured products. It was owing to the discovery of copper and iron ore in the upper peninsula of Michigan in 1844 and the opening of the Sault Ste. Marie canal in 1855 that the great growth of Cleveland ensued, for this city then became the natural meeting place of the bituminous coal from the coal fields of southern Ohio, Pennsylvania and West Virginia, and the iron ore from the Lake Superior district. At Cleveland there are ten blast furnaces with an average output of 120,000 tons monthly, and an exceedingly great variety of manufactured products.

In a paper on, "Modern Methods of Burning Blast Furnace Gas in Stoves and Boilers," Mr. Ambrose N. Diehl gave the results of exhaustive investigations made of stove and boiler operation. The lack of accurate data among those using blast furnace gas made the problem more difficult. In considering the efficiency of a burner it is necessary to distinguish between the performance of the burner alone and that of the unit consisting of the burner and stove. A standard of efficiency which could serve as a basis of comparison was defined as follows: "A gas burner is operated at 100 per cent combustion efficiency when the analysis of a sample drawn from a point 2 ft. beyond the point of ignition, shows perfect combustion." Mr. Diehl's paper was exceptionally complete and the data presented are of considerable value, since the proper utilization of blast-furnace gas with relation to burner construction is an important subject which is receiving more attention. Mr. Diehl's paper was presented in abstract. Burners were classified into six types and their application to hot-blast stoves was first considered and a record of a number of tests was given in detail, with a similar treatment of gas burners for boilers. Most of the tests failed to show data in which the burner was isolated from the boiler or stove so that the data could only serve as a basis for study or adaptation to local conditions.

In the discussion of this paper Mr. H. P. Howland concluded that "for use on stoves we believe the pressure burner to be preferable. Regarding boiler burners, first, the point of automatic regulation as emphasized by Mr. Diehl, does not seem to us to be vitally essential; second, it should be admitted by all that a big step in advance has been taken in equipping a boiler house with burners capable of easy hand regulation in the place of the so-called common burner."

Mr. A. E. Maccoun agreed with Mr. Diehl that there was no type that has so far been developed which might safely be called the most efficient burner, but that only testing out would decide the best type to use.

In a paper on "The Development of Commercial Alloy Steels," Mr. Edgar D. Rogers: reviewed the history of alloy steels and showed how they came into commercial use. He said that Hadfield justly designates the year 1888 as the beginning of a thorough knowledge of alloy steels, when the use of silicon and manganese for the improvement of the physical and magnetic qualities was made practical. The chromium steel industry was founded in America by Julius Baur, and its use has become very extensive. Nickel steel was given an impetus in 1889 by Mr. Riley's paper before the British Iron and Steel Institute, and its use is established and increasing especially in automobile construction. Vanadium for steel was made commercially available when a vast deposit was unearthed in South America in 1905, and its properties are well known, in high-speed and anti-fatigue metal.

As to specifications Mr. Rogers said it had been his experience to receive orders on which certain chemical limits that were commercially possible were outlined, and then physical results demanded that were absolutely impossible with the prescribed analysis. The discrepancies are in a good measure due to the steel makers themselves because only a few years ago the data at hand on alloy steels were obtained through tests in which the type of steel was the only variable. He predicted that the next few years would show a greatly increased demand for alloy steels, and said the electric furnace has enabled the manufacturer to duplicate many steels which were formerly made in the crucible and at a reduced cost.

A paper on "Heat Treatment in Automatic Electric Furnaces" was presented by Mr. Thaddeus F. Baily. Mr. Baily reviewed the development of heat treatment and showed how it became necessary to devise an automatic electric furnace in order to remove the human element as much as possible. He then described a complete equipment and its operation. "The important element in the operation is the temperature measuring devices adopted to take the temperature of the material under treatment itself, not the furnace temperature." These special pyrometers actuate the interlocked electrically operated hydraulic valves when the proper temperature has been reached and discharge the material for quenching.

In the discussion Mr. S. T. Wellman suggested that the mechanism be operated electrically.

Dr. J. W. RICHARDS said that automatic heat treatment eliminating the personal factor of the workman, is a great step in advance. With proper design and heat insulation, electric furnaces of the Baily type should be capable of putting into the articles to be treated 50 per cent of the heating energy of the current. A combined fuel and electric furnace with electric heating for the last 100 or 200 deg. might be much cheaper in some localities. There seems no doubt that some such device would improve rolling practice by discharging ingots from the furnace always at the proper temperature.

Mr. Baily, in replying to the discussion, stated that their furnaces under observation had shown a thermal efficiency of 92½ per cent. In reply to a question by Dr. A. S. Cushman as to the arrangement of pyrometers, Mr. Baily advised placing the tube ½ in. from the metal at the point of discharge, and to Dr. Cushman's statement that surface temperatures were not a correct indication of interior temperatures, Mr. Baily said they allowed such a liberal period for heating that the piece was heated throughout uniformly.

A paper on "Control of Piping and Segregation in Ingots," was presented by Dr. HENRY M. HOWE, explaining the more important elementary principles by which piping and segregation in steel ingots may be lessened, in his characteristic lucid style.

In discussing this paper Mr. E. R. KEENEY said: "Dr. Howe has given us an explanation of many of the features which we have observed in the cooling of ingots, and in the main his reasoning is corroborated by our observations. His statement that during the whole of the preneutral period (i. e. the period during which the outer parts cool faster than the inner) the contraction of the outer parts tends to be more rapid than that of the central ones, is not necessarily true. It would depend on the relative coefficients of contraction of the two parts." Another point on which Mr. Keeney was not convinced was the effect on segregation of the upward currents caused by the gas bubbles in rising steel.

A paper on the "Mechanical Development in Sintering Materials" was presented by Mr. BETHUNE G. KLUGH, showing the intricate problems which were overcome in working out a simple process and describing the progress made in developing the continuous down-draft sintering machine of Dwight and Lloyd. In the discussion Messrs. Robert E. Brooke and Hermann A. Brassert gave their experiences with the Dwight-Lloyd machine. A paper on "Under-Advertising of the Steel Business" was presented by Mr. GEORGE H. JONES.

Non-Ferrous Metal Market

Copper.—The last week in October and the first week in November saw little change in copper prices with only a fair demand. Electrolytic rose slightly on Oct. 28, being quoted at 17.75@18 delivered terms. On Nov. 1 it rose slightly again and producers were asking 18@18.25, which price remained steady during the first week in November. Some authorities think the price was raised to induce buying instead of the usual practice of allowing concessions; in other words, what is known as a bull campaign was thought to be on. There have been 10,000,000 lb. removed from the regular monthly supply on account of the shut down of Greene-Cananea and the strike at Clifton, Ariz. The total exports from Atlantic ports during the month of October were 23,657 tons of 2240 lb. (compiled by N. Y. Metal Exchange). The output of the northern Michigan mines for October was 24,949,000 lb., the largest for any month this year. Exports so far this month have been very low, only 79 tons having been reported up to Nov. 5. Prices quoted on Nov. 5 were: Prime Lake, 18@18.25; Electrolytic, 18@18.25; Casting, 17.50@17.75. Standard in London was still £15 below Electrolytic.

Tin.—The tin market during the last of October and first week in November showed a gain of approximately 3 cents per pound. Quotations of the N. Y. Metal Exchange Oct. 20 for spot were 33.12½@33.50. On the 27th it rose to 33.62½@33.87½. By Nov. 1 it was 35.50@36.50, and on Nov. 6 Straits or Malacca tin was quoted at 35.95@36.37½. During this period the London market showed a steady advance, Straits tin rising from £153 to £162 5s. from Oct. 20 to Nov. 6, with a corresponding advance in Standard. The continued advance in London was thought to have been caused by the scare of shipments being endangered. The talk here of an export duty and the endangering of shipments probably also had its effect. Arrivals during October were only 2000 tons, with arrivals in November up to the 6th, 865 tons. The next two months are normally considered dull ones in volume of consumption.

Lead.—Lead has seen three advances since Oct. 20. On the 21st the Trust price was advanced 0.25 cent to 4.75. On the 29th a 0.15 cent advance was made to 4.90, and on Nov. 4 another advance was made to 5.00. The reasons given for these advances were the continued demand for export, and also an increasing demand for home consumption, the sheet lead trade being reported as very good. Lead was quoted Nov. 5 at 4.92½ at St. Louis, which is the usual differential. The total exports for October were 4106 tons with exports so far in November very low.

Zinc.—Spelter has advanced considerably since Oct. 20, when it was quoted at 13.50@13.75 for spot, with November at 13.00 and December at 12.50. During the last week in October large tonnages were sold for export and the price rose steadily to 14.67½@14.80. Towards the first of November the market became dull, but remained firm, however, and on Nov. 4 the demand became more active with prices at 15.17½@15.42½ spot and futures higher than previously.

Other Metals.—Of the other metals aluminium advanced during the last two weeks and is now quoted 56@58 for Virgin metal. Antimony rose from 30.50 on Oct. 20 to 35@35.50 on Nov. 6. Silver was then quoted at 50, nickel at 45@50, and quicksilver at $100 flask.

The Iron and Steel Market

Spectacular movements are occurring in the iron and steel markets. All precedents are violated and physical conditions, not trade practices or theories as to what is the best for the industry, are making the market from day to day. There are two dominant factors, that steel is extremely scarce, and that there is a heavy demand for special steel, chiefly for war purposes. The unprecedentedly heavy demand for steel as a whole makes it that finished steel products advance rapidly in price, and the commodities advance *pari passu*, because it is not the demand for the particular finished product, but the scarcity of the crude steel, that causes the advance. The large demand for special steel makes it that the ordinarily self contained operations, producing steel for their own consumption, in many cases buy soft steel for their own finishing mills and use their steel making equipment for the production of special steel, chiefly in the form of billets. The Pittsburgh Steel Company has bought about 40,000 tons of Bessemer billets, for use in its wire departments, thus releasing some open-hearth capacity for the manufacture of special steels. Several steel and sheet mill interests have bought sheet bars for their sheet mills and are selling special billets.

The war demand for steel is increasing, and is running to new products, there being offered large orders

for steel sheets, for trench covering and other purposes. The strictly domestic demand has increased sharply, the railroads being the most conspicuous buyers.

The making of contracts for future delivery of finished steel, subject to specifications to be furnished at intervals, has been greatly decreased. Many mills, perhaps the majority, have stopped making contracts entirely and are accepting business only in the form of specific orders.

Finished steel prices have been advancing rapidly, and by an average of not far from $2 a ton in the single fortnight. Unfinished steel is advancing likewise, but close comparisons cannot be made as the market is not well defined. Some consumers are regular customers and are given preference, while there is a sharp divergence between Bessemer and open-hearth steel, and another and still sharper divergence between ordinary soft open-hearth steel and steel that is in any respect special either as to composition or method of rolling.

Pig Iron

In the first few days of November there was a wave of buying of Bessemer and basic pig iron by central western mills, at but slight advances over the market previously current. Subsequent to these purchases the market has advanced rather sharply, the point being reached at which doubts arose whether the production of pig iron by merchant furnaces could be materially increased. In the earlier months of the general market movement there was always in sight a large number of idle merchant furnaces. While steel prices are now as high as at any time since the 1905-6-7 movement, pig iron has not yet reached levels attained in the 1909 and 1912 movements. With the present very sensitive market, however, new price levels may be attained very shortly. The market is now quotable as follows: No. 2 foundry iron, delivered Philadelphia, $16.50 to $16.75; f.o.b. furnace, Buffalo, $16 to $16.50; delivered Cleveland, $16.30; f.o.b. furnace, Chicago, $17 to $17.50; f.o.b. Birmingham, $13 for prompt and $13.50 for first quarter; f.o.b. valley furnaces, 95 cents higher delivered Pittsburgh; Bessemer, $17; basic, $16; foundry and malleable, $15.50 to $16; forge, $15 to $15.50.

Steel

The large purchases of Bessemer billets by the Pittsburgh Steel Company, about 40,000 tons, together with the diversion of much finishing capacity to the use of Bessemer instead of open-hearth steel, has served to make a real and an advanced market for Bessemer billets, at $25.50 to $26.50, maker's mill, Pittsburgh or Youngstown. Bessemer sheet bars are about 50 cents higher. Open-hearth steel is about $1 a ton higher than Bessemer, but the curious condition exists that billets are likely to show a larger spread than this and sheet bars a smaller, peculiar conditions existing as some steel mills have recognized regular customers for sheet bars, while ordinary purchasers of billets do not have recognized sources of supply to the same extent. Ordinary forging billets command $42 to $45, while war billets command $50 to $60. Wire rods are about $35.

Finished Steel

Nearly all finished steel prices have advanced in the past fortnight. Prices now quoted are chiefly for shipment at mill convenience, against specific orders, there being few contracts. Premiums are frequently paid for particularly prompt shipment. Prices quoted are f.o.b. Pittsburgh unless otherwise noted:

Rails, standard sections, 1.25c for Bessemer, 1.34c for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.60c.

Shapes, 1.60c.

Steel bars and bands, 1.60c, base; hoops, 1.75c.

Iron bars, 1.659c to 1.759c, Philadelphia; 1.45c, Chicago; 1.50c to 1.60c, Pittsburgh.

Plain wire, 1.70c; wire nails, $1.85 per keg, base; painted barb wire, 2.00c; galvanized barb wire, 2.70c; fabricated wire fence, 69½ per cent off list, in carload lots.

Sheets, blue annealed, 10 gage, 1.80c to 1.90c; black, 28 gage, 2.25c to 2.35c; galvanized, 28 gage, 3.80c to 3.90c; painted corrugated, 28 gage, 2.45c to 2.50c; galvanized corrugated, 28 gage, 3.85c to 3.95c.

Tin plate, $3.30 for 100-lb. cokes.

Steel pipe, ¾ to 3-in., black, 78 per cent off list; galvanized, 63½ per cent off list.

Boiler tubes (less than carloads) 3½ to 4½-in., 69 per cent off list.

Structural rivets, 2.25c; boiler rivets, 2.35c.

The Chicago Pneumatic Tool Co. has issued a small booklet describing its Simplate flat disc valves which are used in its pneumatic compressors.

The Bristol Company of Waterbury, Conn, was awarded the Grand Prize at the Panama-Pacific Exposition for its exhibit of recording instruments. This exhibit was described in our issue of June, 1915.

The Swenson Evaporator Co., Chicago, Ill., reports that its evaporators are or will be soon installed for producing potash in the United States from all sources, viz., kelp, liquor, natural western lake waters, bittern, feldspar, molasses waste, and alumnite.

Gas Manufacture.—In Technical Paper 120 recently issued by the Bureau of Mines a bibliography of the chemistry of gas manufacture is given. It is intended to serve as a reference guide for those interested in gas making. The authors are W. F. Rittman and M. C. Whitaker.

Aluminium.—It has been reported that the Southern Aluminium Company's plant, work on which was stopped at the beginning of the war on account of the preponderance of foreign capital, has been sold to the Aluminum Company of America. The consideration was said to be about $6,500,000, and work is reported to have been started to finish the plant.

The Simonds Mfg. Co., of Fitchburg, Mass., has signed contracts with the United States Steel Corporation for license of a 5 to 6 gross ton Heroult three-phase electric melting furnace. Results obtained in experimenting and practical use of electric furnaces in the past five years have indicated the desirability of installing the present furnace, which is expected to be under power by January, 1916.

Coal and Mining Laws.—It has been estimated that there are more than 3,500,000,000,000 short tons of coal in the United States, exclusive of Alaska. In mining a comparison with the practices of other countries is helpful. Geological Survey Bulletin 505, "Mining Laws of Australia and New Zealand," contains much suggestive information. This report may be obtained free from the Director of the Geological Survey at Washington, D. C.

Lime Production in 1914.—The total lime manufactured and sold in the United States in 1914 was 3,380,928 short tons, according to the U. S. Geological Survey. This was valued at $1,400,686. Both the quantity and value for 1914 are less than for any previous year since 1908. Lime for chemical works showed a decrease of 20,872 tons from 1913, and a decrease in value of $137,050, owing to a drop of 18 cents in the average price per ton. Both quantity and value, however, exceeded those of 1910, 1911 and 1912.

Thermal Principles of the Blast Furnace

BY J. E. JOHNSON, JR.

(*Continued from page* 792)

Returning now for a moment to the comparison of the theory of the blast furnace with thermodynamics it may be shown that what may be called the availability factor can be expressed in the same mathematical terms in both cases.

The second law of thermodynamics is expressed by the formula $Q \times \frac{T_1 - T_2}{T_1}$, in which Q is the total quantity of heat concerned in the transformation, T_1 is the temperature at which it is applied and T_2 is that at which the unused portion is rejected.

Now in the blast furnace if the whole quantity of heat developed in the hearth be Q, the theoretical combustion temperature be T_1 and the critical temperature be T_2, since the heat given out by a gas in cooling is proportional to the temperature drop $T_1 - T_2$, the portion of the whole which can be utilized in the hearth is $\frac{T_1 - T_2}{T_1} \times Q$, exactly the same as in the case of steam engine.

This is in one sense only a matter of casual interest, but in view of the increasing connection recognized between chemistry and thermodynamics it is highly suggestive. It will be noted that the idea that there are two heat balances for each furnace, one for the shaft, the other for the hearth and bosh, is set forth very plainly in the quotation before given from "Notes on the Physical Action of the Blast Furnace," but the difficulty of getting at the debit side of the account seemed almost insuperable, and on this account I did not for several years use the chart of hearth heat for any but comparative purposes to show the heat obtainable per pound of coke with different blast and critical temperatures and different percentages of moisture in the blast, because the furnaces of which I had the management were coke furnaces on relatively lean ores, and in such cases the hearth heat obtainable per pound of coke exerts exclusive control of the economy of the furnace when other details of operation receive proper attention.

A few years ago when I undertook the management of a charcoal furnace using rich ores I attempted to apply the same principles and methods of calculation to it, but soon found that there were many features of its operation which could not be explained on that basis.

For instance, covering in the ends of the blowing cylinder and piping the air to it from out of doors in a very dry climate yielded no better results than taking the air from the engine room, although figures from the chart of hearth heat based on the humidity in the engine room indicated that a considerable saving should result. Similarly, changes of blast temperature exerted little or no influence on the running of the furnace compared with what similar changes would do on a coke furnace, and in particular it seemed impossible to save the furnace by the use of more blast heat when it started to run cold. It was absolutely necessary to put more fuel down into the hearth.

Finally it occurred to me that the hearth heat per pound of iron generated in that practice might be greater than was needed, and that a comparison of that developed in charcoal practice with that developed in coke practice might be illuminating. I had been taking observations of the free-running temperature of the slag, so I knew approximately what it was; this, with the blast temperature and the fixed carbon in the charcoal per pound of iron, supplied the necessary data for the calculation, which showed 1300 B.t.u. of hearth heat developed per pound of iron, while that developed in coke practice at the Isabella furnaces was only about 1000 B.t.u.

This astonishing result was the more remarkable because about 1200 lb. of slag per ton of iron were made in the coke practice and only about 500 in the charcoal. This could only be interpreted in one way, that more hearth heat was being developed than the furnace needed, and the excess was passing up into the shaft of the furnace and finding utilization there, but only in doing relatively low-temperature work. This in turn could only mean that the heat developed in the shaft was deficient in comparison with the hearth heat, and that if means could be found to increase the shaft heat the increase would prepare for final smelting in the hearth a greater proportion of ore, of which the excess of hearth heat could take care, so that the burden could be increased and the fuel consumption reduced, and incidentally, as we shall later see, the quality of the iron improved.

The ore used was a clayey Gogebic ore carrying normally 13-14 per cent of moisture, and a preliminary calculation showed that the abstraction of the heat required to evaporate this moisture from the small volume of gas which a charcoal furnace produces must lower the temperature of the gas about 400 deg., and as it was being discharged at about 200 deg. Fahr. it seemed as though the gas temperature would be high enough to do considerably more reduction, especially as the gas was amply rich for this purpose, if it did not have to drive off this water.

This seemed then to be a case in which the fuel economy of the furnace was controlled exclusively by the shaft heat developed per pound of fuel, the hearth heat playing a purely secondary rôle.

In order to ascertain whether this could be true detailed calculations were made covering the operation of the furnace as it was running and as it was believed it could run with dry ore. A similar set of calculations was then made covering the operation of a coke furnace which had made an excellent run to see how the existing and assumed charcoal cases compared with it.

The results of the whole investigation, which covered several months of laboratory work and all my time which could be spared from other work for three months or more, were embodied in a report to the executive of the company.

The major portion of this report is given here, as it covers the situation better than can be done in any other way.

THE EFFECT OF WET ORE ON CHARCOAL-FURNACE PRACTICE

"The conception that the great quantity of moisture present in the soft ores used in the charcoal practice of this district might affect the working of the furnace by undue cooling of the top gradually took shape in my mind from observing the top temperatures at a time when our fuel consumption was high.

"According to all previous experience high fuel goes with a high top temperature, but last fall when we were having fuel consumptions of 95 to 100 bushels our top temperature was down around 200 to 250 deg.

"A brief calculation based on the volume of gas and the quantity of moisture in the ore showed that the heat required to evaporate the water was sufficient to lower the temperature of the gas 300 to 400 deg. We had also found, by a good many analyses, that our top gases were as rich in CO as those of a coke furnace, although burning as we do much less fuel per ton of iron than in coke practice this should not be the case. An analysis

of the conditions was begun, based on three considerations:

"1. The ability of the gas to carry off all the oxygen of the ore with a given fuel consumption. This depends on the extent to which CO can be oxidized to CO_2 in the presence of iron ore at different temperatures.

"2. The total amount of heat developed in the furnace by the oxidation of the fuel to the extent at which it passes out in the top gases.

"3. The development of a sufficient amount of heat in the hearth to carry on the reactions which can only take place above the temperature at which the cinder runs freely.

"The importance of this last point rises from the following consideration: The fuel, say at the free-running temperature of the cinder, is burned by the blast at its incoming temperature with the development of a certain amount of heat, which, in the absence of burden, would heat the gases to a temperature known as the theoretical combustion temperature. When the burden is present, these gases are cooled down by it and give out their heat as they cool, but it is obvious that they cannot supply any heat at or above the free-running temperature of the slag (known as the critical temperature) after they are cooled down to that temperature. The amount of heat which they give out in cooling from the theoretical combustion temperature to the critical temperature may therefore be called the hearth or high-temperature heat. In coke practice the amount of this heat developed is generally smaller proportionately to the needs of the furnace than the total heat developed, and, consequently, it is the amount of this heat available which limits the fuel economy of a coke furnace. But in charcoal practice the critical temperature, that is, the free-running temperature of the slag, is so much lower than that of coke furnaces that the range from theoretical combustion temperature to critical temperature is much greater, and, therefore, the amount of hearth heat developed per pound of carbon is much higher, and it has been found by calculating the amount of high temperature heat developed per hundred pounds of iron in good coke practice, and the amount developed in our practice, that the latter was considerably *higher* than the former, although the slag volume to be melted and other work to be done in the hearth is much lower in our case.

"This indicated to my mind that we were developing more hearth heat than we needed, and making our iron hotter and of poorer quality in consequence, and I thought at that time that we should have some way of increasing the oxidation of the top gases so that we could increase the amount of heat in the upper part of the furnace, which, for lack of a better term, I call the shaft heat—shaft being the only name I know of for the portion of the furnace from the bosh to the top.

"After I had made the calculation showing how much the temperature of the gas was lowered by having to evaporate the moisture in the ore, it seemed to me that if the ore were thoroughly dried the top temperature would go up so much that the gases would be oxidized a good deal further, which would develop more heat per pound of carbon in the shaft, and as we had a surplus in the hearth this would enable more burden to be carried.

"In order to make reasonably accurate calculations, it was necessary to investigate along three lines as a preliminary: (1) the free-running temperature of the cinder; (2) the exact composition of the charcoal; (3) the extent to which the CO could be oxidized in the presence of iron ore at given temperatures.

"1. We have been carrying on, to the best of our ability, investigations of temperature of the running iron and cinder for over two years, and while they have not been completely satisfactory, they seem to indicate a temperature of 2300 deg. Fahr. as that at which a normal charcoal slag will run sufficiently freely for the furnace to work. Similar observations on coke furnaces several years ago indicated a temperature of about 2700 deg. Fahr. as that of a good slag in Bessemer practice.

"2. The composition of the charcoal. This was a matter on which we had made experiments in the past, but they were not successful enough to be considered reliable. We, therefore, put a piece of charcoal in an alundum boat in a silica tube heated by an electric furnace, surrounded it with an atmosphere of CO and heated it until no more gas came off. The gas was caught in rubber bags, its volume measured and its analysis obtained. The volume of CO used to fill the furnace was deducted from the total volume and the analysis of the gas was figured back to what it would have been if the CO had not been added in advance. Such a determination was made for birch and one for maple. The results checked exceedingly closely. As a further check the specific gravity of the gas was taken by weighing a small gas bag full of it, and this was compared with the specific gravity calculated from its analysis. The check was excellent. The charcoal remaining was weighed and then burned. The weight as taken from the electric furnace less the weight of the ash was considered to be the fixed carbon of the coal. From the analysis and volume of the gas evolved the weight of each gaseous component was figured and these were reduced to percentages by weight of the original sample of charcoal. The variations between birch and maple were very slight, and were averaged to obtain the figures given in Table IV.

TABLE IV.—AVERAGE CHARCOAL ANALYSIS FOR BIRCH AND MAPLE

	Per Cent
CO_2	4.50
O_2	0.67
CO	11.10
H_2	1.65
CH_4	3.40
N_2	7.77
Fixed carbon	68.50
Ash	1.00

"3. The extent to which the CO could be oxidized in the presence of iron ore at given temperatures. After overcoming the worst of the difficulties in this determination, we were unable from lack of time to get a long series of these determinations at different temperatures, as we had hoped, but we demonstrated by several runs that at a temperature of about 700 deg. Fahr. 62 per cent of CO_2 were in equilibrium with 38 per cent of CO. This means that if sufficient time were allowed it would be possible to oxidize the top gases so that the CO_2 would hold this ratio to the CO at this temperature. We also proved conclusively that the equilibrium percentage was much higher in CO_2 as the temperature arose. At 800 deg. it is about 70 per cent to 75 per cent, and at about 900 deg. it is around 80 per cent CO_2, but these figures are not exact.

"These were the data on which our calculations had to be based. It was decided first to take the conditions which prevailed with this furnace at the time these calculations were started. Ninety-five bushels of coal, 50 per cent ore yield, 13.50 per cent moisture in the ore, 400 lb. of limestone per ton of iron. From the tonnage made and the revolutions of the blowing engine the amount of blast per hundred pounds of iron was obtained, and by temperature observations of the air going into the cylinder and coming out it was possible to get a rough check on the displacement of the air cylinder. All gas volumes are reduced to 30 in. barom-

eter and 32 deg. Fahr. The specific volumes of gas are those given in Kent, and heats of combustion are taken from the same source, but the specific heats of the different components in the operation are taken from Richard's "Metallurgical Calculations." The specific heats given in Kent are not exact for high temperatures.

"It is necessary to call attention to one fact of much importance. The charcoal gas contains appreciable quantities of hydrogen and methane. Hydrogen requires for its combustion eight times its weight of oxygen, and methane for combustion to water and CO_2 requires four times its weight of oxygen. These are both very active reducing agents and will take up oxygen from the ore with great rapidity at a reasonably high temperature, 600 to 800 deg., but with the conditions under which the furnace works the top temperature is so low that they pass off largely without action on the ore. Our analyses have repeatedly proven them to be present in top gases. The quantity of oxygen which they can absorb is so great that the question whether they are oxidized completely or not affects the composition of the top gas tremendously.

"I have assumed that in a furnace using wet ore and 95 bushels of coal three-fourths of the hydrogen was burned and none of the methane, because this gives results which check out well with the actual gas analyses we obtained from the furnace, but, unfortunately, we did not get any during the exact period covered by the above assumption. However, the analysis of the gas in both CO_2 and CO, as well as in hydrogen and methane, is so close to what we actually found that no material error can be introduced into the heat calculations by any inaccuracy in these assumptions.

"A further fact is that the heat developed by combustion of CO and that developed by hydrogen per pound of oxygen consumed is nearly the same. Therefore, it is a matter of slight importance in regard to the development of heat whether the oxygen to be disposed of is consumed by hydrogen or by CO.

"Having made the calculation for 95 bushels of coal, I then made an exactly similar one assuming 75 bushels of coal with the ore thoroughly dried, and in this calculation I assumed that the gases would pass off at 400 deg. higher temperature than in the other, 632 deg. as against 232 deg., assumed as a fair average of the actual temperatures for 95 bushels with wet ore. This assumption was made because of the fact before mentioned that the amount of moisture present in the ore is sufficient to cool down the gases about 400 deg. In this case I assumed that the hydrogen and methane were completely oxidized because previous investigation on a furnace with a moderately warm top had shown this to be the case.

"In order to get some comparative figures from another source, I have made a similar analysis of the run of a coke furnace described by A. K. Reese, in Vol. 27 of the *Transactions* of the American Institute of Mining Engineers. This was a six months' run on the furnace at Canal Dover, Ohio. In order to check out the quantity of wind blown with the gas analysis given, it was necessary to assume that the volatile matter in the coke in this latter case consisted of methane and hydrogen, and that these were oxidized in the top of the furnace, as above described.

"Before proceeding to discussion of the results obtained in the three cases, it will be well to point out that the method of calculation of the total heat developed in the furnace is the time-tried and accepted one. The method of calculating the heat in the hearth has been used for the last seven or eight years, and has proven itself so reliable in giving relative figures in comparative cases that it is now generally accepted. See METALLURGICAL AND CHEMICAL ENGINEERING, March, 1913, page 147-8, "The Gayley Dry Blast," by H. M. Howe.

"In regard to the degree of oxidation of the gases in any given case, it is not commonly realized that this can be figured with considerable accuracy. The amount of blast blown supplies a given quantity of oxygen and the ore charged supplies another quantity. A slight additional amount comes from the gas given off by the charcoal, and a somewhat larger amount from the CO_2 of the limestone. All these have to come out of the stack, combined with the carbon which goes in in the form of fuel and in the form of CO_2 in the limestone, except for that which comes out combined with the hydrogen of the fuel. This, of course, is in the form of water and disappears in the gas analysis.

"As we know, with considerable accuracy, all of these weights of carbon, and hydrogen, and of oxygen, the degree of oxidation of the products of combustion can be accurately calculated and the heat developed determined with corresponding accuracy.

"All of the ores in this district are hematites, and therefore each seventy units of iron contain thirty units of oxygen, and the oxygen combined with the iron can be accurately figured from this. To this must be added the oxygen which comes from the silicon going into the iron, and due account must be taken of the fact that iron contains only about ninety-five units of Fe. I have assumed about 1.00 per cent to 1.25 per cent silicon, and have neglected the manganese in all cases, as the amount of oxygen accompanying it is too small in proportion to the possible variations in the larger quantities to be given serious consideration. This gives 42 lb. of oxygen released from the ore and silica per hundred pounds of iron. (It is very desirable to have a symbol which shall represent pig iron as distinguished from chemically pure iron, Fe, and for this purpose I have used the letter "I," because iodine does not enter into the discussion and no confusion can result.)

"The volume of blast blown was estimated from the displacement of the blowing cylinder, some allowances being made for the suction in the cylinder when the valves close and for the re-expansion of the compressed air into the clearance space, both of which tend to reduce the actual pounds of air blown. The temperature of the air being discharged from the cylinder was taken at the discharge head, and simultaneously that of the outside air, with which the engine was supplied. The heating of the discharge air was about 20 deg. higher than the theoretical for the pressure blown at the time, 6 lb., and this was assumed to take place before compression began and to make a further reduction of the weight of air delivered. The weight of air blown, derived from the displacement, with these corrections and with slight allowances for the loss in changing stoves and leakage, was used in the gas calculations, and checked out very satisfactorily with the actual analysis of top gas found. It may be well to remark here in passing that if incorrect assumptions be made, the calculated top gas will vary widely from the actual top gas analysis, and will at once indicate that something is wrong.

"It will be noticed that in the tables the weight of fixed carbon in the fuel per hundred pound of iron is given, and from this is made a deduction of 4 lb. to correspond with the 3.50 per cent or 4.00 per cent of carbon combined with the iron and some slight loss of solid carbon at cast and at flush time.

"The result of the calculations are each represented on a separate table, Tables V, VI and VII, in which it has been attempted to epitomize the calculation and to indicate the various reactions that lead to the top gas as finally discharged. In a general way the table gives,

FIG. 5

COKE PRACTICE

1667 lbs., of Coke Per Ton of Iron
(Canal Dover, Ohio, 1897)

			HEARTH and BOSH												SHAFT																	
COMPONENT	Line Nos.	SOURCE	In Hearth and Bosh	Product	Disposal	Temp. at Ent'nce	Heat per lb. at Temp. of Ent'nce	Heat Brot to Hearth per 100 lb. Iron	Heat Comb per lb.	Heat of Comb per 100 lb. Iron	Total Heat in Hearth	Temp. at Exit	Heat per lb. at Temp Exit	Heat Taken Out of Hearth B.t.u.	At Top of Bosh per 100 lb. Iron	Released in Shaft	Before Re-Action	Re-Action	Product in Top Gas Lbs N	CO	CO2	H2	CH4	H2O	Gas Total	Heat from 32 to 432° per lb.	Heat Car out Top	Heat of Comb per lb.	Heat Comb per 100 lb. Iron	Heat Brot in at 2700	Total Hts. in Sh'ft	Cu. Ft.	Vol% of Top Gas
		Column Letters	A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S	T	U	V	W	X	Y	Z	AA	BB	CC	DD	EE
O	1	From Blast	76.			1100°	238.5	18200			18200																					
C	2	Burnt by O from Blast	57.						4450	253800	253800																					
	3	C O Produced		133.								2700°	729.	96950																		
	4	C for I & lost in Hearth	4.																													
	5	Total of C used in Hearth	61.			2700°	1120.	68300			68300																					
	6	Total fixed C in Furnace	61.																													
	7	C burnt in Shaft with O from Ore	—																													
	8	C O Produced	—																													
N	9	N from Blast	254.5												254.5																	
	10	N from Fuel																														
	11	Total N	254.5			1100°	272.	69200			69200	2700°	725.	184600					254.5						254.5	96.3	24500			184600	184600	22.50	63.7
H2	12	H2 from Fuel														0.55																
	13	H2 Burnt														0.55												61500	23800		23800		
	14	O for H2 Burnt															4.4															
	15	H2O produced																						4.95								
	16	H2 remaining																				—			—							
CH4	17	CH4 from Fuel														2.2												23500	49500		49500		
	18	O for CH4 to H2O + CO2															8.80															
	19	Product CO2																			6.05											
H2O	20	H2O Produced																						4.95								
	21	H2O from Ore																						10.80								
	22	Total H2O																							20.70	1200	26500						
O	23	O from Fe + Si														42.0																
	24	O from Fuel																														
	25	Total O															42.0															
	26	Deduct O for CH4 + H2 Lines 18 & 14															13.2															
	27	Deduct O for fixed C not burnt by Blast															—															
	28	Net O for CO															28.8															
CO	29	CO from fixed C													133.														96800	96800		
	30	CO from fuel gas														—																
	31	Total CO															133.0															
	32	CO oxydized by net O, Line 28															50.4												4350	219200	219200		
	33	CO remaining															82.8			82.8					82.8	96.8	8140				1665	29.7
CO2	34	CO2 from CO + O																			79.2											
	35	CO2 from Fuel gas																														
	36	CO2 from Stone																			12.8											
	37	Total CO2																			98.05				82.05	57.0	8525				795	26.6

Total 409600
Removed by CO + N 281400
" " Dissoc H2O in Blast 9300
" " H2 from " " " 1800
" Total 292500
Net Heat used in Hearth 117080

281400

Total 68300 482800
Heat Removed by Top Gases 68300
Net Heat used in Shaft 514500

FIG. 6

CHARCOAL PRACTICE

HEARTH and BOSH — Wet Ore, Actual Case — 95 Bushels Coal, Per Ton Iron — SHAFT

COMPONENT	Line Nos.	SOURCE	In Hearth and Bosh	Product	Disposal	Temp. at Ent'nce	Heat per lb. at Temp. of Ent'nce	Heat Brot to Hearth per 100 lb. Iron	Heat Comb per lb.	Heat of Comb per 100 lb. Iron	Total Heat in Hearth	Temp. at Exit	Heat per lb. at Temp. Exit	Heat Taken Out of Hearth B.t.u.	At Top of Bosh per 100 lb. Iron	Released in Shaft	Before Re-action	Re-action	Product in Top Gas Lbs. N	CO	CO2	H2	CH4	H2O	Gas Totals	Heat from 32 to 212° per lb.	Heat Car out Top	Heat of Comb per lb.	Heat Comb per 100 lb. Iron	Heat Brot in at 2300°	Total Hts in Sh'ft	Cu. Ft.	Vol % of Top Gas
		Column Letters	A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S	T	U	V	W	X	Y	Z	AA	BB	CC	DD	EE
O	1	From Blast	62.1			1000°	212.	13200			13200																						
	2	Burnt by O from Blast	46.6						4650	207000	207000																						
C	3	C O Produced		166.7								2300°	866.	96200																		
	4	C for I & lost in Hearth	4.																														
	5	Total of C used in Hearth	50.6			2300°	922.	46000			46000																						
	6	Total Fixed C in Furnace	58.1																														
	7	C burnt in shaft with O from Ore	7.5																														
	8	C O produced		17.5																													
N	9	N from Blast	297.9			1000°	343.5	50000			50000	2300°	806.	125700	297.9															125700	125700		
	10	N from Fuel														6.6																	
	11	Total N															214.5	→	214.5						214.5	47.2	10100					2740	58.8
H2	12	H2 from Fuel														1.4																	
	13	H2 Burnt														1.05												61500	64000		64000		
	14	O for H2 Burnt															8.4																
	15	H2O produced																→						9.45									
	16	H2 remaining														0.35						0.35			.35	.600	200					62	1.3
CH4	17	CH4 from fuel														2.89		→					2.89		2.89	82.5	240					65	1.4
	18	O for CH4 to H2O + CO2															—																
	19	Product CO2															—				—												
	20	H2O produced																						—									
H2O	21	H2O from Ore																						27.0									
	22	Total H2O																							36.45	1100	42300						
O	23	O from Fe + Si														42.																	
	24	O from Fuel														.57																	
	25	Total O															42.57																
	26	Deduct O for CH4 + H2 Lines 18 & 14															8.4																
	27	Deduct O for fixed C not burnt by blast															10.0																
	28	Net C for CO															24.17																
CO	29	CO from fixed C													166.7	17.5														96200	96200		
	30	CO from fuel gas														9.3																	
	31	Total CO															135.5																
	32	CO oxydized by net O, Line 29															42.3												4350	182000		182000		
	33	CO remaining															93.2			93.2					93.2	47.3						1192	25.5
CO2	34	CO2 from CO + O																→			66.5						4600						
	35	CO2 from fuel gas																			3.5												
	36	CO2 from stone																			8.0												
	37	Total CO2																			78.3				78.3	40.5	3170					635	13.5
							Heat Removed by CO + N				317400			101900											425.7	86610				440400	4695		
							Net Heat Used in Hearth				125500														39.45	Heat Removed by Top Gases				60000			
																									380.2	Net Heat Used in Shaft				379800			

FIG. 7

CHARCOAL PRACTICE

HEARTH and BOSH — Dry Ore, 75 Bushels of Coal, Per Ton of Iron, Estimated Results — SHAFT

COMPONENT	Line Nos.	SOURCE	In Hearth and Bosh	Product	Disposal	Temp. at Ent'nce	Heat per lb. at Temp. of Ent'nce	Heat Brot to Hearth per 100 lb. Iron	Heat Comb per lb.	Heat of Comb per 100 lb. Iron	Total Heat in Hearth	Temp. at Exit	Heat per lb. at Temp Exit	Heat Taken Out of Hearth B.t.u.	At top of Bosh per 100 lb. Iron	Released in Shaft	Before Re-Action	Re-Action	Product in Top Gas Lbs. N	CO	CO2	H2	CH4	H2O	Gas Totals	Heat from 32 to 632° per lb.	Heat Car out Top	Heat of Comb per lb.	Heat Comb per 100 lb. Iron	Heat Brot in at 2330°	Total Hts in Sh'ft	Cu. Ft.	Vol % of Top Gas
O			A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S	T	U	V	W	X	Y	Z	AA	BB	CC	DD	EE
	1	From Blast	56.			1000°	212.	11900			11900																					
	2	Burnt by O from Blast	62.						4459	186800	186800																					
	3	C O produced		98.0								2360°	608.0	59650																		
	4	C for I & lost in Hearth	4.																													
	5	Total of C used in Hearth	66.			2300°	922.	42400			42400																					
C	6	Total Fixed C in Furnace	66.																													
	7	C burnt in shaft with O from Ore	—																													
	8	C O Produced		—																												
	9	N from Blast	187.5			1000°	243.5	45000			45000	2360°	600.	113600	187.5															113600	113600		
N	10	N from Fuel														5.2																
	11	Total N															192.7	→	192.7						192.7	149.	28700					2460	62.3
	12	H2 from Fuel														1.1																
	13	H2 burnt														1.1													61500	67700		67700		
H2	14	O for H2 Burnt															8.8																
	15	H2O produced																						9.9									
	16	H2 remaining														—						—			—								
	17	CH4 from Fuel														2.3									—			23500			54150		
CH4	18	O for CH4 to H2O + CO2															9.2																
	19	Product CO2																			6.33												
	20	H2O produced																						5.17									
H2O	21	H2O from Ore																						—									
	22	Total H2O																							15.1	1380	20800						
	23	O from Fe + Si														42.																	
	24	O from Fuel														0.44																	
	25	Total C															42.44																
O	26	Deduct O for CH4 + H2 Lines 18 & 14															18.00																
	27	Deduct O for fixed C not burnt by Blast															—																
	28	Net C for CO															24.44																
	29	CO from fixed C													98.															59000	59000		
	30	CO from Fuel Gas														7.4																	
CO	31	Total CO															105.4																
	32	CO oxydized by net O, Line 28															42.9												4350	136500		180900		
	33	CO remaining															62.6	→		62.6					62.6								
	34	CO2 from CO + O																			67.2					149.5	9400					903	20.4
CO2	35	CO2 from Fuel Gas																			3.												
	36	CO2 from stone																			8.												
	37	Total CO2																			84.5				84.5	136.	11500					685	17.3
								Heat Removed by CO + N			236700			173200												70400				251600	3947	
								Net Heat Used in Hearth			113500															Heat Removed by Top Gas				70400			
																										Net Heat Used in Shaft				111200		

in its upper left-hand corner, the details of what goes on in the hearth and bosh, and works gradually toward the final result in gas composition and heat development at the right-hand end and bottom of the table. It is feared that these charts may not be perfectly comprehensible at first glance, but no effort has been spared to make them as logical and clear as possible, so as to save the reader an enormous amount of effort in going over written details of these calculations.

"Coming now to the final results in heat development, we find that Table V, giving the actual conditions with wet ore, shows a development of hearth heat of 125,500 B.t.u. and a development of shaft heat of 379,800 B.t.u., a total per pound of iron of 505,300 B.t.u.

"The coke furnace in which, by the way, the theoretical gas analysis checks remarkably well with the actual results, shows a development of hearth heat of 117,000 B.t.u. and of shaft heat of 515,600 B.t.u., a total of 632,600 B.t.u. The assumed case, Table VII, dry ore and 75 bushels of coal, shows a development of hearth heat of 113,500 B.t.u., of shaft heat of 411,200 B.t.u., a total of 524,700 B.t.u. These are arranged in condensed tabular form below:

	Coke Practice, 1664 Lb. Coke	Actual Practice, Wet Ore, 95 Bu. Coal	Assumed Case, Dry Ore, 75 Bu. Coal
Hearth heat	117,000 B.t.u.	125,500 B.t.u.	113,500 B.t.u.
Shaft heat	515,600 B.t.u.	379,800 B.t.u.	411,200 B.t.u.
Total heat	632,600 B.t.u.	505,300 B.t.u.	524,700 B.t.u.

"Before proceeding to a discussion of these figures, it should be stated that in the case of the coke furnace it was assumed that the air blown contained four-tenths of a pound of moisture per thousand cubic feet of air, all of which moisture was dissociated in the hearth with corresponding absorption of heat. Further, that the hydrogen evolved by the dissociation of the water passed out of the top of the bosh at the same temperature as the other gases and abstracted a corresponding amount of heat from the hearth. The dissociation absorbs 9300 thermal units and the hydrogen absorbs 1800, a total of 11,100 thermal units. This assumption as to the amount of moisture present is based on previous experience of the moisture in engine houses during the winter months. It is not possible that the air was taken from outside the engine room, as no mention is made of that fact and it was not customary at that time (1879).

"To be strictly accurate, allowance for the oxygen from this source should be made in the gas calculations, and as the hydrogen generally recombines in the top of the furnace with oxygen from the ore, the 11,100 thermal units subtracted from the hearth heat might, with propriety, be added to the shaft heat. These corrections have not been made because their magnitude is not of sufficient importance to affect the general result, and they would greatly complicate the tables to show them. No allowance for moisture is made for our practice because we take our air from out of doors and the amount of moisture present in the air in winter time in this climate is a negligible quantity.

"Comparing first the two actual cases, coke and 95 bushels of charcoal, we see that the heat developed in the hearth with the charcoal is considerably greater, while the heat developed in the shaft is very much less than with coke. The difference in the latter condition of over 100,000 thermal units is very striking. The explanation of the major part of the difference lies in higher temperature to which the materials must be heated in the shaft of the coke furnace than in the charcoal, and also the much greater amount of limestone to be decarbonated in the coke than in the charcoal. Only approximate calculations are possible on this subject for lack of accurate data as to specific heats of the materials at these high temperatures, but such calculations indicate that most of the difference can readily be accounted for in this way.

"Turning now to the high temperature, or hearth heat, the conditions are extraordinary. The coke furnace, with a considerably greater quantity of slag, greater radiation losses and every disadvantage compared with a charcoal furnace, develops about 7 per cent less high-temperature heat than the charcoal furnace. Moreover, an examination of the table will show that while 58 lb. of fixed carbon are brought into the furnace by the charcoal, only about 50 lb. are consumed in the hearth—4 by absorption in the iron and 46 by combustion with the blast. This shows that about 8 lb. out of 58, or 14 per cent, of the total fixed carbon are dissolved by the oxygen of the ore. If this carbon came down into the hearth as it should, the excess of high-temperature heat for the charcoal furnace would be very much greater than it is.

"It seems well to point out here one curious fact. When CO is oxidized to CO_2 by oxygen taken from the ore, heat is given out by the reaction; but when carbon is burnt to CO by oxygen from the ore, an enormous amount of heat is *absorbed* for every pound of oxygen which is transferred from the ore to the carbon. This is because a unit of oxygen develops over twice as much heat in burning CO to CO_2 as it does in burning C to CO. Knowing the amount of carbon which is thus burnt directly, we can easily figure the heat absorbed. This amounts to 33,000 thermal units per 100 lb. of iron. We do not know where this reaction takes place, but the action of this furnace and the frequent observation of forge cinder at or below the level of the tuyeres when the furnace is in normal operation causes the belief that a great portion of this combustion of fixed carbon with oxygen from the ore takes place in the hearth itself, and thereby robs the hearth of the excess heat which it develops.

"Extensive investigations have been made in Germany on the equilibrium between FeO and Fe_3O_4 with CO and CO_2. These investigations proved that at temperatures of 1800 deg. to 2000 deg. CO could contain a considerable percentage of CO_2 and still deprive FeO of all its oxygen; but if the temperature went higher than this the affinity of the iron for oxygen increased so much that pure CO or incandescent carbon would be required to deoxidize it completely. This means that at the top of the bosh where the combustion of the blast to CO is complete the temperature should not be too high, as high temperature prevents complete deoxidation by the CO, and requires the use of solid carbon for that purpose, with consequent waste of fuel. Thus it may actually be that excess fuel leading to high temperature in the hearth exercises an unfavorable action on the final reduction of the ore.

"It is perfectly obvious that the effect of wet ore on gas is to cool it down with extreme rapidity as it approaches the top of the furnace, and while the physical heat thus lost is of some importance, a condition of vastly greater importance results from it. This is that the temperature of the gas is so far reduced that it can no longer react on the ore. This means that oxygen which should be removed from the ore near the top of the furnace with decided evolution of heat descends to lower levels, and that which should be reduced in the central zone descends to or above the top of the bosh before it is finally removed. This involves the reaction with solid carbon, above pointed out, and the destruction of fuel economy.

"The effect of the excess heat in the hearth I believe to be bad also for the quality of the iron, because in the region around the tuyeres the temperature developed is excessive, this temperature being given up by the

gas to the stock in a slightly higher zone, in order to overcome the cooling effect produced through the direct reaction of the oxygen of the ore by the incandescent fuel. These high temperatures at the tuyeres tend strongly to superheat the iron as it drips down through this zone. This increases its affinity for carbon, which the presence of an excess quantity of carbon permits it to satisfy. Therefore, the tendency of this working is to produce a hot-made high-carbon iron, although the silicon may not be high, due to the scouring action of the oxide which comes down to, or nearly to, the zone of the tuyeres.

"Turning now to the assumed case of correct top conditions and a fuel consumption of 75 bushels of coal, we find that the heat developed in the hearth is the same, or virtually the same, as that in the coke furnace (although the requirements as above pointed out are not so great), while the heat developed in the shaft of the furnace is very materially greater than with 95 bushels of coal and wet ore, about 407,000 thermal units against 378,000.

"The critical point in this assumption is the ratio of the top gases. It will be seen that this works out to 17 per cent of CO_2 and 20.3 per cent of CO, as compared with the actual results in the coke furnace of 15.6 per cent of CO_2 and 20.7 per cent of CO, a ratio of 0.75 : 1 for the coke furnace to 0.84 : 1 for the assumed charcoal conditions. This is not a very much higher ratio of CO_2, and is far lower than the equilibrium ratio, which is 1.63 : 1; that is, 62 CO_2 to 38 CO. In other words, it is possible for ore to oxidize the carbon gas to a very much higher degree than would be required for 75 bushels of coal under the given conditions.

"In regard to the time allowed for this reaction, it is well to note that the volume of the furnace in proportion to the volume of gas passing through it is very much greater in the charcoal furnace than in the coke one, which allows a correspondingly longer time for reaction.

"The development of more heat in the shaft of the furnace with over 20 per cent less fuel than in the actual case may appear to indicate erroneous figures, but this is not the case if we admit the possibility of the degree of oxidation just explained. The amount of oxygen from the ore for a hundred pounds of iron is virtually constant. For the same analysis of iron and same kind of ore it is exactly constant.

"This constant quantity of oxygen would oxidize the same quantity of CO to CO_2 in either case and develop the same amount of heat in doing so if all the fixed carbon were burnt to CO by the blast, but the smaller the quantity of fuel burned in the hearth per pound of iron, the greater the proportion of this which is oxidized to CO_2, and therefore the greater development of heat per pound of fuel. Moreover, in cases where all the carbon is not burnt to CO in the hearth by the blast (the case of wet ore, for instance), a part of this oxygen from the ore expended in burning C to CO, which, as explained above, generates less than one-half the heat per pound of oxygen consumed that is generated by the oxidation of CO to CO_2, hence not only a less proportion of the total CO is oxidized in the case of the larger fuel consumption, but the total supply of oxygen available for this purpose is reduced about one-fourth to satisfy the fixed carbon of the fuel, hence only three-fourths as much is available for oxidation of CO to CO_2 as in the case of smaller fuel consumption. Thus not only the relative but also the absolute oxidation of CO to CO_2 is much smaller with the larger fuel consumption. This, of course, is only true within the limits of possible oxidation of carbon gas above described.

"It might be thought that this whole line of argument was fallacious because it was the hearth heat which limited the fuel economy, as it undoubtedly is in most coke practice. If this were so we could produce an enormous change in the burden-carrying capacity by changing the temperature of the blast. The heat brought in by the blast for 95 bushels of coal at 800 deg. is 50,000 thermal units, at 1000 deg., as given in the table, 63,700, and at 1200 deg. 78,700. The high-temperature heat being 124,800 B.t.u. for 1000 deg. blast temperature would become 111,100 for 800 deg. and 138,800 B.t.u. for 1200 deg.; therefore, raising or lowering the blast temperature 200 deg. would produce a difference of about 11 per cent either way, or there would be an economy with 1200 deg. of blast temperature over that at 800 deg. of about 24 per cent. As a matter of fact, the difference which we find in fuel consumption for such a change in blast temperature is about 5 per cent. This corresponds approximately to the difference in total heat supplied to the furnace in the two cases, which figures about 5.50 per cent.

"It may be taken as established, therefore, that the amount of hearth-heat produced is not the limiting condition in regular charcoal practice, but that this limitation comes from the amount of shaft heat developed per pound of carbon burnt. This depends, as already pointed out, on the degree to which the top gases can be oxidized by the ore, and this, in its turn, depends on the temperature of the gases. If they remain hot they will continue to react on the ore until they are discharged, but it would be as unreasonable to expect them to do this when chilled with water and diluted with steam as to expect a fire to burn as freely with a hose turned on it as without."

As a result of this investigation arrangements were made to dry 500 tons of ore for a trial run, and while this amount was sufficient for only two days the results bore out in the most striking manner the conclusions derived from the calculations. The fuel consumption was decreased about 12 per cent, and this threw the deficiency of heat into the hearth, with the result that the furnace responded to changes in blast temperature to an extent that was almost incredible, in view of previous experience, and everything indicated, as far as so short a trial could, the complete agreement of practice with this theory of two thermal equations instead of a single one, as in the older theory, and that both of these must be satisfied before the furnace would function, even though the satisfaction of one involved a great surplus of heat above the requirements for the other.

A second article on the thermal principles of the blast furnace will follow.

Electroplating with Cobalt.—The complete report on the work done on plating with cobalt by Dr. Herbert T. Kalmus, C. H. Harper and W. L. Savell has been issued by the Canadian Department of Mines as Bulletin No. 334. A description of this work was given in our issue of May, 1915.

The Mineral Resources of Texas, by William B. Phillips, is the title of Bulletin No. 365 of the University of Texas. The work is restricted to mineral products and undeveloped resources are not considered. It is a very complete account of the products of Texas and contains much valuable information.

Products and By-Products of Coal.—In Bulletin No. 323 of the Canadian Department of Mines, recently published, Messrs. Edgar Stansfield and F. E. Carter present a general outline of the products and by-products obtained from the utilization of fuels in Canada. The publication is of special interest in view of its connection with the dyestuff problem which has confronted Canada as well as the United States.

Potash from Wood and Plant Ashes

BY HARLOW BRADLEY

INTRODUCTION

Recently several companies have been formed for the extraction of potash from the enormous kelp beds along the Pacific Coast, but whether this method will be commercially successful or not remains to be seen.[1]

Considerable interest has been evinced, however, since the European war stopped imports from Germany, in the possibility of producing potash from wood ashes, and the present paper will attempt to deal with this phase of the subject.

HISTORICAL

The production of potassium carbonate, or potash, from wood ashes is one of the oldest of the purely chemical industries and was fairly extensive fifty years ago in Russia, the United States and Canada. Elm, birch, larch and maple were the woods most generally used, and the devastation of the forests in the vicinity of leaching plants was beginning to assume alarming proportions.

The rapid development of the well-known Stassfurt potash deposits after 1870, however, had an immediate effect on the production of potash from wood ashes. In twenty years the amount produced in this way was not even half of the world's production, as the figures for 1890, given in Table I, furnished by H. Grüneberg,[2] will show:

TABLE I—WORLD'S PRODUCTION OF POTASH IN 1890

	Tons
Wood ashes—Russia, Canada, United States, Hungary, Galicia	20,000
Beet sugar waste—France, Belgium, Germany	12,000
Mineral salts—Germany, France, England	15,000
Suint (wool scourings)—Germany, France, Belgium, Austria	1,000
World's production	48,000

"These conditions differ strikingly from those which existed thirty years ago (in 1860), when wood ash was in exclusive use, and Russia potash ruled the market."

The industry in America was slowly undermined until, at the present time, the amount of "pearl ash," or refined "potashes," made either in the United States or Canada, is insignificant. Ashes are still used for fertilizer to a small extent, and farmers often lixiviate them to produce a lye for soft soap; an industry still exists for making a commercial potash, but it is very small. Before the war practically all of our potash was coming from the Stassfurt mines, giving the German Kali Syndicate a virtual monopoly.

The Twelfth Census[3] gives some interesting figures on the extent of the potash industry in America fifteen years ago.

"Though returns for the census year 1900 were received from sixty-seven establishments, producing 3,864,766 lb. of potash, valued at $178,180, no pearl ash was reported manufactured. Of these sixty-seven establishments, twelve produced products valued at less than $500. Cognizance was taken of this industry in the reports of the United States as early as 1850, so that the data are at command for comparing the condition of the industry in this country for each decade since 1850, as set forth in Table II.

"This Table II shows that there was a constant decrease in the value of the product, though the quantity varied somewhat."

[1] See Bulletin No. 248, Univ. Cal. Agric. College: J. S. Bird, Economic Value of Pacific Coast Kelps.

[2] Wagner—Chemical Technology, 1892, p. 299.

[3] Twelfth Census of the United States, Vol. X—Part IV—pp. 543-546.

TABLE II—TOTAL PRODUCTION OF POTASHES, BY DECADES: 1850 TO 1900

Year	Number of Establishments	Product		Average Price per Pound Cents
		Pounds	Dollars	
1850	560	.....	1,401,533	....
1860	212	.....	538,550	....
1870	105	.....	327,671	....
1880	68	4,571,671	232,643	5.09
1890	75	5,106,939	197,507	3.86
1900	67	3,864,766	178,180	4.82

The establishments reported in 1900 were distributed as stated in Table III:

TABLE III—GEOGRAPHICAL DISTRIBUTION OF POTASH FACTORIES—1900

States	Number of Establishments	Average Number of Wage Earners	Capital	Value of Product	Per Cent of Total
United States	67	.	$70,899	$178,180	100.0
Michigan	44	52	25,851	79,642	44.7
Ohio	15	25	20,050	35,519	19.9
Indiana	3	4	2,275	6,560	3.7
Maine, Wisconsin, Illinois	5	11	22,723	56,459	31.7

"There were reported as having been used in this manufacture 812,399 bushels of wood ashes valued at $40,191. The yield of potash per bushel of ashes, as reported, varied from 2.4 to 7 lb. In the product given above there is included potash packed in cans, amounting to 820,000 lb., having a value of $53,349. Excluding this, as being in the nature of a duplication, it appears that the total production of potash for 1900 was 3,044,766 lb., and that, therefore, the average yield of potash per bushel of wood ashes, as shown by the extra returns, was 3.75 lb. Pelonze and Fremy[4] give the yield by weight as 10 per cent, and this appears in other textbooks; but all returns for ashes received at the census of 1900 were given in bushels.

"From the data for production given above it is evident that although the average price of potash for 1900 was higher than for 1890, the industry was not remunerative and consequently the total quantity and value of the product decreased.

"The cost of producing a barrel of 650 lb. of potash was stated by a Michigan manufacturer to be as given in Table IV:

TABLE IV

Ashes, 150 bushels, at 3c	$4.50
Hauling ashes	6.00
Fuel	2.00
Labor	3.00
Barrel, cost of	1.25
Repairs, interest, etc	1.50
Total	$18.25
Selling price at works	25.00
Gross profit, per bbl	$6.75

"The ashes, therefore, yielded 4 1/3 lb. of potash per bushel, and the potash sold at 3.85 cents per pound. It will be noted that the weight of a barrel of potash is given above as 650 lb. From the returns it appears that the net weight of a barrel of this material varied from 650 to 740 lb., the average being about 700 lb."

USES OF POTASH

The chief use of potash is in the form of crude mineral salts for fertilizer purposes. Over $13,000,000 worth of these crude salts was imported in 1912. Large quantities of potassium in the form of the cyanide are used in the recovery of gold from tailings and low-grade

[4] Traité de Chemie, 1865—Vol. II, p. 225.

ores. The glass industries require potash for certain forms of glass, and some potash is used for the manufacture of soft soap. Previously most of the manufactured potassium salts were imported, but potash is now used in this country for the manufacture of the chromate, the hydrate (caustic potash) and several others.

MANUFACTURE OF POTASH FROM WOOD ASHES

Potash is prepared by dissolving out the soluble contents of wood ashes and evaporating the solution to dryness. The process as carried out formerly on a commercial scale is as follows:

The wood, often mixed with vine stalks and ferns, is incinerated in pits 3 or 4 ft. deep and sheltered from the wind. The material is thrown in in portions and burned until the pit is nearly full of ashes. The latter are then removed, spread on a floor and dampened. According to an old observation ash which has been stored a long time produces more potash. This is because the silicate of potassium is transformed to carbonate by the carbon dioxide and water in the air.

The ash should be spread in moderately thin layers and turned for several months. It is doubtful, however, whether the increased yield of potash pays for the large amount of floor space necessary. Should the silicate content of the ash be comparatively high, an artificial method could probably be introduced. Often (in American practice) a small amount of lime (5 per cent) is added.

The ash is now placed in tubs or vats, formed from tar barrels by cutting them in half. These are arranged in double rows, one above the other, to permit the weaker leachings to run from the upper vats to the lower. The vats are provided with two cross beams, which support a cullendered false bottom covered with straw, and below this is a cock for the removal of the lye. Water, preferably hot, is poured on the mass in the upper vats and emerges strong lye.

If it is 32 deg. Tw. strong (20 deg. Bé., 20 per cent to 25 per cent salt), it can be boiled down at once; the weaker liquors and washings, however, are run through several vats successively. Care should be taken to have the ash well tamped down in the vat. Using cold water would leave the potassium sulphate behind, but hot water is used to obtain more potash, and the sulphate can be separated out later. The latter is valuable to the manufacturers of alum.

The leached residue, which contains some potash and available phosphoric acid, makes a fair fertilizer. The lye is boiled down in shallow evaporating pans made of iron with corrugated bottoms to produce a greater heating surface. The boiling down is generally a rough process. The waste heat can be used to warm the feeder which supplies the pans. In lieu of pans possessing a large heating surface (pans fired from the top would be best), circular cast-iron pots are often employed.

Evaporation progresses rapidly at first and new supplies of strong liquor are poured in, the heat being continued until a sirupy consistence is attained, when the fire is gradually slackened, and the contents of the pan becoming solid (containing about 6 per cent moisture) are dug out and placed aside as crude potashes.

In spite of much stirring before the mass becomes quite dry, much of it burns fast to the pan, causing a great waste of fuel and a speedy destruction of the bottom.

In Russia the lye was concentrated in copper kettles to the point of crystallization and then discharged into wooden cisterns where potassium carbonate crystallized out. After draining, the carbonate was calcined and the mother liquor returned to the cistern.

In America the mass was heated nearly to redness and converted into greyish-pink compact potashes, which were assorted into firsts, seconds, thirds and unbrandables.

The brown crude potashes are calcined in a reverberatory furnace, which causes the carbonaceous and other organic matter to burn away, the hydrate being carbonated and the sulphide converted into sulphate. The loss in weight during this operation is from 10 to 20 per cent. Too great a heat is to be avoided because the mass would then fuse, leaving a mixture of fine charcoal, from the carbonaceous material, and salt which cannot be burned white.

Potassium carbonate is difficult of fusion, but when large amounts of chloride are present the mixture fuses at an incipient red heat. The resulting product is white with a bluish tinge, the latter due to small amounts of potassium permanganate, contains more carbonic acid than the crude product, and is called "pearlash." A purer material is obtained by dissolving this pearlash in water, separating the insoluble matter, evaporating the liquor, and calcining it again. This latter process was generally used in America.

The American practice used so much lime that the major portion of the potassium existed as hydrate. In this way some of the silicate, which would otherwise have been lost, was recovered as hydrate. Potash with a large quantity of lime in it must be even more carefully protected against attracting atmospheric moisture than ordinary commercial carbonate, although the latter is also deliquescent. Such semi-caustic potash was very much in demand by soap makers, as it took less lime to make their lye and caused less loss.

In purifying carbonate of potash either of two methods can be employed: The carbonate may be stirred up in an iron pot with 1½ or 2 times its weight of cold water, and the mixture, which gets hot, allowed to cool down before siphoning off the clear liquor. The sulphate and some chloride remain behind together with the insoluble matter; the residue is not washed (which would cause the sulphate to be redissolved), but only pressed. The liquor, containing mostly carbonate, but also chloride and silicate, is brought to dryness, stirring it up to the end.

The second method is to dissolve the material in hot water; the residue is exhausted in the same way. The liquors are boiled down to a specific gravity of 1.50 and allowed to cool, when the sulphate and some chloride crystallize out. This process is more expensive than the former.

In order to make still purer carbonate for flint glass, pearlashes are calcined with a little sawdust, lixiviated, and the clear liquor boiled down and calcined. This is repeated twice over, and the last product is dried down until it is converted into granular "crystallized" carbonate with 15 per cent to 18 per cent water (less deliquescent—$K_2CO_3 \cdot 2H_2O$).

Fig. 1 is a sketch of a typical calcining furnace. Air entering through *b* passes up through grate *a* and the flame plays over *t* upon the calcining hearth. The furnace should be warmed before charging and should be run continuously.

When the furnace is red hot 300 or 500 lb. of crude potash are broken up and spread over the calcining hearth. Water begins to come off, violently if too much is present, so that the heat must be raised gradually. It takes about one hour to drive off the moisture and constant agitation must be practised.

The heat is then increased to redness so that the combustible and coloring matter is consumed. At this period the potash takes fire, burning with a flame of slight intensity, becoming black from the carbonization

of the combustible material. It becomes clearer and clearer until at length a sample drawn off appears quite white. Some potash sticks to the hearth of the furnace, and the flame is allowed to play over it until it softens, when it can be removed. The white calcined ash is removed and allowed to cool, and should be packed in close casks before it has an opportunity to absorb moisture.

POSSIBLE IMPROVEMENTS IN THE METHOD OF MANUFACTURE

Assuming that a sufficient supply of wood ashes was available to supply a plant for operation at the present time, many improvements could be made in the leaching, evaporating and calcining apparatus over the old methods. Leaching vats with a greater capacity than the tar barrels, and provided with cullendered, metal, false bottoms and a more efficient filtering medium than

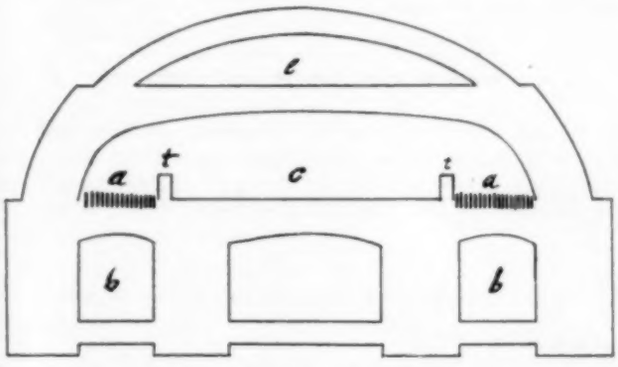

FIG. 1—CALCINING FURNACE

aa—grates. bb—ash pits. c—calcining hearth 3 to 4 ft. wide. tt—fire bridges, iron plates. e—not necessary, may be used for other purposes.

straw, could be arranged in batteries much after the fashion of beet sugar diffusion batteries. In place of the shallow evaporating pans, or the cast-iron pots, an evaporator of the Enderlein type provided with a rotary calcining furnace, could be used, such as is employed in the recovery of soda at the soda and sulphate pulp mills. Fuel being no object, since the ashes would be a source of potash, this arrangement could do without the vacuum evaporators used at pulp mills for the preliminary concentration of the weak spent liquor, thus saving a considerable investment. A brief description of the Enderlein evaporator follows.

This apparatus consists essentially of an open wrought-iron vessel about 16 ft. long, 7 ft. deep and 14 ft. wide, inclosed in a brick chamber. Two shafts, each mounted with thirty-two wrought-iron disks, 5 ft. 6 in. in diameter, set about 5 in. apart, run across the vessel. The shafts are mounted 7 ft. apart, center to center, and the disks are intended to dip into the liquor. About one-sixth of their surface is submerged in the lye, giving a total surface exposed to the hot gases of 3140 sq. ft. The shafts revolve at nine or ten revolutions per minute.

At the front of the evaporating chamber, and opening into it is the rotary calcining furnace or roaster. This is about 16 ft. long by 8 ft. in diameter, slightly inclined away from the evaporator, and revolves at a slow rate of speed dependent on the conditions of evaporation. The liquor, which runs continuously into the evaporator, runs out in concentrated form through a weir into the rotary roaster, where it is evaporated nearly to dryness, and drops out at the other end between the mouth of the roaster and the smelter.

The smelter as used at pulp mills is merely a firebrick lined chamber or dutch oven, but for recovering potash a calcining furnace of the hearth type might be used. As the black ash drops out at the end of the roaster it is shoveled into the calcining furnace where the organic matter burns, providing most of the heat for evaporation.

About 15,000 to 20,000 gallons of waste soda pulp liquor can be evaporated from 16 deg. Bé. to 37 deg. Bé. with this apparatus in twenty-four hours, or about 1.25 lb. of water per square foot of heating surface per hour. The operation of this apparatus would probably have to be modified somewhat in dealing with potash.

CHARACTERISTICS OF POTASH PRODUCED FROM WOOD ASHES

Calcined potash is a hard, light, porous, and granular saline mass—the white color merges into pearl grey or becomes yellowish or bluish. Single pieces often exhibit blue and red spots on the fractured surface. The red is caused by oxide of iron, grey by carbon particles, and blue by the action of the alkali on oxide of manganese, giving rise to small amounts of potassium permanganate. Insoluble matter in varying quantities is generally present in commercial potashes—partly due to imperfect filtration through the straw in the leaching vats and partly to ashes being carried over in the calcining furnace.

In the soluble portion potassium carbonate, potassium sulphate, potassium chloride, and potassium silicate are always present. Sodium carbonate, potassium phosphate, potassium bicarbonate, potassium hydrate, sulphide of potassium, manganate of potassium, and organic matter are sometimes present.

In the insoluble portion there is always silicic acid present, and sometimes silicate and phosphate of lime, calcium carbonate, magnesium carbonate, magnesium phosphate, oxide of iron, protoxide of manganese, alumina, and lastly sand and carbon.

The caustic potash present is due to the addition of slaked lime, as stated before, and is found only in American potash. Sulphide of potash is caused by the action of organic matters on the sulphate, while bicar-

TABLE V—ANALYSES OF COMMERCIAL POTASHES AND PEARLASHES, PRODUCED FROM WOOD

Origin	Quality	K_2CO_3 Per Cent	KOH Per Cent	Total Alkali Calc. as K_2CO_3	Total Potash Calc. as K_2CO_3	Na_2CO_3 Per Cent	K_2SO_4 Per Cent	KCl Per Cent	Analyst
American potash...	.....	68	.....	75.5	68	5.8	15.3	8.1	Pesier
American potash...	1	41.7	49.6	108.2	106.4	1.4	4.0	2.0	F. Mayer
American potash...	2	19.4	44.4	87.8	77.2	8.2	16.1	5.6	F. Mayer
American potash...	2	18.3	36.6	81.7	65.9	12.2	15.1	14.6	Grüneberg
American potash...	2	34.5	29.6	81.8	77.9	3.0	15.0	7.5	Grüneberg
Montreal potash....	Firsts	43.87	50.03	92.49	88.83	6.24	16.07	4.41	Tatlock
Montreal potash....	Firsts	26.16	36.50	89.82	85.54	2.84	10.40	1.68	Tatlock
Montreal potash....	Seconds	38.53	42.44	85.79	76.24	11.47	18.92	12.82	Tatlock
Montreal potash....	Seconds	21.71	30.63	85.08	73.98	8.52	9.18	6.13	Tatlock
Montreal potash....	Thirds	46.31	6.14	72.10	53.86	17.81	20.53	7.63	Tatlock
Canadian pearlash..	.....	71.3	.....	74.2	71.3	2.3	14.3	3.6	Pesier
Canadian pearlash..	.....	65.0	.....	70.2	65.0	4.0	21.0	7.0	Grüneberg
Canadian pearlash..	1	77.50	.....	80.98	77.50	2.86	11.65	2.65	Tatlock
Canadian pearlash..	.....	71.38	.....	74.53	71.38	2.31	14.38	3.64	Tatlock

bonate of potassium must be due to the action of carbon dioxide from the gases.

Table V shows the composition of various potashes produced in America.

THE PRACTICAL UTILIZATION OF ASHES

THE INORGANIC CONSTITUENTS OF PLANT AND WOOD ASHES

Plant ashes of all kinds, as well as wood ashes, contain varying quantities of potassium salts; in fact, many herbaceous plants contain much higher percentages of this material than trees do.

Sage brush (artemesia), for example, which grows in considerable quantity in the West, contains from 5 to 7 per cent of K_2O, based on the weight of the plant, or about 20 to 30 per cent on the weight of the ash.[5] The brush must be cleared from the land to enable plowing, and this is often done by dragging heavy railroad rails over the ground, the brush being collected in piles and burned. It would not be profitable to save the ashes for leaching in this case, although they can be thrown back on the land for fertilizer.

In this connection it might be well to note that when black sage is steam distilled, an oil is produced containing considerable camphor.[6] Should the distillation of this material prove commercially possible, the waste could be burned for its potash content. That the artemesias have been considered as a source of potash is shown by the following quotation from Knapp's Chemical Technology:

"Hermbstädt planned to plant wormwood (an artemesia) for the production of potash, as it grows on very poor soil. One Magdeburg acre (18,000 sq. ft.) affords in a summer three crops, or 20,000 lb. of the dry plant, which produces 2364 lb. of ash, or 936.6 lb. of potashes. The plan was forbidden by the German Government as wormwood takes the potash out faster than it is produced by disintegration of the soil."

This artemesia is not the same as our black sage, which is perennial and would require two or three years' growth between each harvesting.

In Java the stems and leaves of the indigo plant were used to produce potash after the extraction of the coloring matter, while the Russians produce a potash for soap making from the ash of the stalks, hulls and leaves of the sunflower.[7] About one-fifth of this should be thrown back on the land for fertilizer. The sunflower industry in Russia is a large one, the seed being pressed for its oil, or eaten as peanuts are in America.[8] Cottonseed hulls were burned extensively for fuel in the Southern States, and the ashes used for fertilizer or for making potash, but at present the hulls command a higher price as cattle food. An analysis of cottonseed hull ashes is given in Table X. The mangrove tree, growing in marshes where the water is soft, contains high percentages of potash, it is stated.

Potash occurs mainly in the stems and leaves of plants. The amount present varies greatly in different parts of the plant, and is dependent upon the season of the year and the nature of the soil. Two trees of the same species, growing side by side, generally show differences in the amount of potash taken up, so that the ashes found on the market were exceedingly variable in composition.

Tables VI and VII illustrate the above facts very well.

Bottinger showed that the soil may affect the constituents of the ash. A sickly fir tree growing in the neighborhood of a manganese mine contained 15 per cent of protoxide of manganese on the weight of the ash, while in healthy wood only 2.7 per cent was found. In a like manner, Saussure got 12 per cent ash from flowering vetch growing on arable land, while from the same plant grown in distilled water only 4 per cent was found.

Prof. E. Emmons, from 150 analyses of ashes of American woods, deduced the following:

1. Proportion of ash in limbs and branches greater than in inside wood, and greater also in outside than inside wood, though exceptions occur to latter inference.
2. Inorganic matter acquires maximum proportion in bark of trunk.
3. Alkalies acquire maximum proportion in fruits and envelopes.
4. Phosphates acquire maximum proportion in outside woods, fruits and envelopes.
5. Lime acquires maximum proportion in bark.
6. Phosphates and alkalies acquire maximum proportion in bark of trunk.

TABLE VI—VARIABILITY OF PLANT ASHES
Ashes of Fir from Different Localities (Saussure)

	1 Per Cent	2 Per Cent
Carbonate of potash	3.6	7.36
Carbonate of lime	46.34	51.19
Carbonate of magnesium	6.77	None

TABLE VII—AMOUNT OF ASH IN DIFFERENT PARTS OF TREES (SAUSSURE)

	Per Cent Ash
Oak leaves, May 10	5.2
Oak leaves, September 27	5.5
Oak wood	0.2
Oak bark	6.0
Oak wood (branches—sapwood)	0.4

The character of the unleached wood ashes found in the trade is indicated in Table VIII,[9] showing the mean, maximum, and minimum results of the analysis of ninety-seven samples by Grossman.[10]

TABLE VIII—MEAN COMPOSITION OF WOOD ASHES

	Mean, Per Cent	Maximums, Per Cent	Minims, Per Cent
Potash	5.5	10.2	2.5
Phos. acid	1.9	4.0	0.3
Lime	34.3	50.9	18.0
Magnesia	3.5	7.5	2.3
Insoluble	12.9	27.9	2.1
Moisture	12.0	28.6	0.7
CO_2 and undetermined	29.9	....	....

The potassium found in wood ashes is extracted from the soil by the plant during its growth, the presence of potassium compounds in the soil being essential to the growth of vegetation. Consequently, wood ashes are valuable fertilizing material. Ordinary soils contain from 3500 to 12,000 lb. of potash per acre to a depth of 1 ft.

The results of many analyses of plants and plant ashes, partly of European origin, are compiled in tabular form in Tables IX and X.

It is not intended that such materials as rye, straw, potato stems, thistle, and vine pith are to be considered as a practical source for potash. These analyses are merely included for the sake of completeness. The cost of collecting such materials or even their ashes would be prohibitive.

The potash in cottonseed hull ashes varied from 10 per cent to 42 per cent and the total phosphoric acid

[5] See U. S. Patent 896,168, Aug. 18, 1908.
[6] See Bulletin 235, Bureau of Plant Industry, U. S. Dept. of Agriculture.
[7] Chem. Tr. J., Aug. 12, 1911.
[8] Consular Report, No. 137—1892.
[9] Handbook of Experiment Station Work—A Digest. U. S. Dept. of Agriculture, 1893.
[10] Annual Report, Mass. Agric. Exp. Station, 1888—p. 202.

TABLE IX—ASH IN 100 PARTS—PLANTS

Potato stems	15.00 Berthier
Potato leaves	1.15 Mollerat
Oat straw	5.10 Boussingault
Nettle	10.67 Pertius
Thistle	4.03 Pertius
Wheat straw	7.00 Boussingault
Wheat straw	4.30 Saussure
Rye straw	0.30 Karsten
Rye straw	3.60 Boussingault
Corn stalks	12.20 Saussure
Cane stem	1.70 Karsten
Fern	2.90 Mollerat
Fern	5.00 Pertius
Acacia	2.46 Abbene
Grape stalk	8.89 Abbene
Vine pith	7.29 Blengini
Vine pith	3.57 Blengini
Sunflower stalks and leaves	1.93 Wiley
Sunflower stalks and leaves	5.72 Ludogovsky

TABLE X—AVERAGE ANALYSIS OF COTTON SEED HULL ASHES

	Per Cent
Moisture	7.80
Potash	22.75
Sol. phos. acid	1.25
Reverted phos. acid	6.50
Lime	9.60
Magnesia	10.75

from 3 per cent to 13.5 per cent. Some silicate of potassium was always present.

The analyses of woods and wood ashes in Tables XI and XII have been compiled from various sources.[11]

The two analyses given on the top of the right-hand column of this page show the effect of leaching on the composition of wood ashes.

	Per Cent					
	Moisture	Phosphoric Acid	Potash	$CaCO_3$	Insol.	Undeter.
Wood ashes, leached	30.00	1.40	1.1	51.00	13.00	3.5
Wood ashes, unleached	12.00	1.90	5.5	61.00	13.00	6.6

SOME SOURCES OF WOOD ASHES

As a possible source of wood for potash purposes we may turn to our far Western sawmills. Improved sawmill and logging machinery, together with the many more or less well-developed industries using wood waste as a raw material, are doing much to lessen the wastes in the lumber business, but at best a sawmill recovers a scant 40 per cent of the total weight of wood in lumber, lath and shingles. The sawdust, slabs, trimmings and edgings are sold for fuel where there is a market for the material and the rest is burned or used for filling in tide flats and marshes. Within a radius of one small town in the State of Washington there are at least twenty-five sawmills, each of which turns out an average of 150,000 ft. B. M. per ten hour day, and this is excluding the shingle mills.

The total annual output of lumber in this small area is then approximately 1,000,000,000 board feet, or

TABLE XI—ANALYSES OF VARIOUS WOODS AND THEIR ASHES

Material	Air-Dry Wood Contains:			Ash Contains:			
	Ash, per Cent	Phosphoric Acid, per Cent	Potash, per Cent	Phosphoric Acid, per Cent	Potash, per Cent	Lime, per Cent	Magnesia, per Cent
Alder wood	0.375	.....	.....	.....	.....	.....	.....
Alder wood	1.390	.....	.....	.....	.....	.....	.....
Ash wood	0.32	0.012	0.149	3.58	46.04	23.57	0.60
Ash wood	2.30	.....	.....	.....	.....	.....	.....
Ash wood	1.22	.....	0.074	.....	.....	.....	.....
Beech wood	0.58	.....	0.145	.....	.....	.....	.....
Beech bark	.....	.....	0.153	.....	.....	.....	.....
Beech red, wood	0.40	.....	.....	.....	.....	.....	.....
Beech red, wood	0.612	.....	.....	.....	.....	.....	.....
Birch wood	0.275	.....	.....	.....	.....	.....	.....
Birch wood	1.030	.....	.....	.....	.....	.....	.....
Birch charcoal	0.800	.....	.....	.....	.....	.....	.....
Boxwood wood	.....	.....	0.226	.....	.....	.....	.....
Cedar wood	.....	.....	.....	.....	4.81	.....	.....
Chestnut wood	0.16	0.011	0.029	6.76	18.10	49.18	2.11
Chestnut bark	3.51	0.014	0.278	3.25	7.93	47.02	0.01
Dogwood wood	0.68	0.057	0.190	8.51	28.04	38.93	6.80
Dogwood bark	9.87	0.140	0.341	1.42	3.46	49.20	1.40
Elm wood	2.55	.....	0.390	.....	.....	.....	.....
Elm wood	2.28	.....	.....	.....	.....	.....	.....
Eucalyptus wood	0.25	.....	0.024	.....	10.00	.....	.....
Ferns	3.64	.....	0.626	.....	.....	.....	.....
Fir wood	0.23	.....	.....	.....	.....	.....	.....
Fir bark	1.78	.....	.....	.....	.....	.....	.....
Fir charcoal	1.44	.....	.....	.....	.....	.....	.....
Fir needles	2.90	.....	.....	.....	.....	.....	.....
Hickory bark	3.97	0.061	0.141	1.54	3.56	46.82	2.59
Hickory wood	1.050	.....	0.241	.....	.....	.....	.....
Magnolia wood	0.36	0.032	0.071	8.75	19.54	38.94	8.05
Magnolia bark	2.98	0.095	0.192	5.31	11.87	23.64	4.89
Maple bark	9.49	0.421	1.197	4.44	12.61	37.91	3.25
Nettle	.....	.....	2.503	.....	.....	.....	.....
Oak, post, wood	0.77	0.070	0.169	9.00	21.92	46.39	6.88
Oak, post, bark	12.10	0.116	0.249	0.96	2.06	52.04	0.65
Oak, red	6.29	0.103	0.179	1.63	2.84	50.51	.....
Oak, red, wood	0.57	0.060	0.140	10.55	24.66	48.26	5.38
Oak, white, bark	5.95	0.074	0.125	1.24	2.10	52.73	1.62
Oak, white, wood	0.26	0.025	0.106	9.48	42.16	29.85	3.43
Oak, white, wood	1.33	.....	0.153	.....	.....	.....	.....
Oak, white, wood	0.815	.....	0.130	.....	.....	.....	.....
Oak leaves	4.70	.....	.....	3.35	3.74	29.03	.....
Pine, burr, wood	1.09	.....	.....	3.31	6.92	10.30	.....
Pine, wood	0.34	.....	0.045	.....	.....	.....	.....
Pine, Georgia, wood	0.033	0.012	0.050	3.82	15.35	55.24	6.25
Pine, Georgia, bark	0.037	0.013	0.024	1.99	3.56	34.14	2.45
Pine, Oilfield, bark	1.94	0.095	0.077	4.88	3.96	27.95	3.10
Pine, Oilfield, wood	0.18	0.007	0.008	4.11	3.85	67.73	6.54
Pine, straw, mixed	1.65	.....	.....	4.28	2.08	14.47	.....
Pine, yellow, wood	0.23	0.010	0.045	4.18	19.70	65.53	3.20
Pine, black, wood	0.21	0.009	0.030	4.33	14.30	58.98	0.50
Poplar, wood	.....	.....	0.075	.....	.....	.....	.....
Poplar, wood	1.306	.....	.....	.....	.....	.....	.....
Poplar, wood	0.800	.....	.....	.....	.....	.....	.....
Sycamore, wood	0.99	0.121	0.230	12.23	23.17	31.62	0.62
Thistles	.....	.....	0.500	.....	.....	.....	.....
Vetch	.....	.....	2.750	.....	.....	.....	.....
Vines	.....	.....	0.550	.....	.....	.....	.....
Willow	3.40	.....	0.285	.....	.....	.....	.....
Wormwood	2.80	.....	7.300	.....	.....	.....	.....

[11] Karsten, Berthier, Mollerat, Saussure, Winkler, Hertwig, etc.

TABLE XII—ANALYSES OF WOOD ASHES—BERTHIER*

Source of Ashes	Soluble in Water							Insoluble in Water							
	Carbonic Acid, per Cent	Sulphuric Acid, per Cent	Muriatic Acid, per Cent	Silicic Acid, per Cent	Potash per Cent	Soda per Cent	Total per Cent	Carbonic Acid, per Cent	Phosphoric Acid, per Cent	Silicic Acid, per Cent	Lime per Cent	Magnesia, per Cent	Oxide of Iron, per Cent	Oxide of Manganese, per Cent	Total per Cent
White beech wood	.....	.....	.....	.....	.....	.....	19.22	26.92	8.11	4.05	31.31	6.33	1.30	2.76	80.78
White beech charcoal	4.43	1.30	0.83	0.18	9.12	2.14	18.00	24.43	7.22	3.20	35.75	5.70	0.08	5.70	82.08
Red beech, wood	3.65	1.19	0.85	0.16	10.45		16.30	27.53	4.77	4.85	35.66	5.86	1.25	3.77	83.70
Oak wood	2.88	0.97	0.01	0.02	8.11		12.00	34.99	0.71	3.36	48.41	0.53	.....	.....	88.00
Oak charcoal	4.39	0.90	0.62	0.15	9.43		15.50	26.91	6.27	1.52	39.95	7.15	0.09	2.60	84.50
Oak, dark	1.45	0.37	0.04	0.05	4.33		6.25	37.22	.....	1.03	47.78	0.75	.....	6.98	93.75
Lime wood	2.96	0.81	0.19	0.17	6.55		10.80	35.75	2.51	1.80	46.53	1.97	0.09	0.54	89.20
Birch wood	2.72	0.37	0.03	0.16	12.72		16.00	26.04	3.61	4.62	43.85	2.52	0.42	2.94	84.00
Alder wood	.....	1.24	0.06	.....	.....		18.80	25.17	6.25	4.06	40.76	2.03	2.92	.....	81.20
Fir wood	7.34	3.75	.....	1.09	15.32	22.55	50.05	10.75	0.90	6.50	13.60	4.35	11.15	2.75	50.00
Pine wood	2.89	1.67	0.92	0.18	4.41	3.53	13.60	32.77	0.91	4.19	38.51	9.56	0.09	0.36	86.40
Mulberry wood	5.82	2.09	1.01	.....	13.16	2.91	25.00	31.75	1.36	2.19	34.85	3.48	0.38	0.98	75.00
Elder wood	7.71	2.06	0.13	0.06	21.54		31.50	22.06	5.83	2.25	34.57	1.76	0.08	1.26	68.50
Fern	4.35	1.62	3.19	.....	19.84		29.00	17.96	5.68	15.48	30.39	0.50	0.50	0.50	71.10

*Retabulated from Knapp's Chemical Technology.

roughly 1,500,000 tons, assuming the low figure of 3000 lb. per 1000 ft. as the average weight of green lumber. If the manufactured lumber represents even 50 per cent of the weight of the logs and half the waste is sold for fuel purposes there is 750,000 tons of actual waste produced in this locality annually, a possible source for 1000 tons of excellent potassium carbonate, worth $85,000 to $100,000 at ordinary market rates and over $400,000 at the present rate (August, 1915).

Undoubtedly there is plenty of wood being burned by the Western sawmills to produce several times this quantity of potash. As the burners are now operated a large share of the ash is lost, due to the draft. To recover all the ashes a downcomer, together with some means for separating the ashes from the swiftly moving gases, such as a dust catcher, would have to be provided. Difficulties of a like nature have been overcome with blast furnaces and smelters, so that no serious obstacles should be encountered.

The sensible heat of the waste gases could be used for preheating the liquor where the leaching and burning operations were carried on at the same place. With properly designed burners located at the various mills the ash could be collected and sent to a leaching plant without excessive transportation charges.

It might be possible to recover considerable ash in addition from the brush and slashings which are often piled and burned in the logging operations and in the clearing of logged-off land. Several lumber companies in Washington have estimated that there are 40 cords of wood in the average acre of stumps and waste after logging operations. The waste in this part of the country would consist essentially of Douglas fir, Western hemlock and Sitka spruce. Should water freight rates between southern California and Washington ports be sufficiently low it might be possible to combine wood ashes with those produced from kelp at the seashore, thus eliminating several leaching plants and permitting production on a larger scale.

Hardwood distillation plants use considerable quantities of wood and charcoal for fuel, the ash of which is available for potash manufacture in districts where a number of plants are in close proximity. Hardwood ashes are richer in potassium than those from softwood.

Where cottonseed hulls are burned for fuel or any quantity is thrown to waste, they should be considered as a source of potash. It is hardly necessary to state, however, that the demand for potash in this country could in no way be supplied from these sources.

Peat.—According to the U. S. Geological Survey the total production of peat in the United States was 47,093 short tons.

Barometric Condensers.—The Mesta Machine Co., Pittsburgh, Pa., has issued Bulletin R, describing its barometric condensers for vacuum installations.

The Dow Chemical Company, Midland, Mich., the American manufacturer producing indigo, has increased its capital stock from $1,500,000 to $3,000,000, the additional stock being distributed as a dividend.

Erecting Acid Plants.—One indication of the unusual activity in the sulphuric acid industry is the fact that the Chemical Construction Company, Charlotte, N. C., have contracts for the erection of several concentrating plants, six Glover towers and six Gay-Lussac towers. This equipment consists of acid-proof brick and cement with no lead casing around the towers.

Canadian Salt Industry.—In Bulletin No. 325 of the Canadian Department of Mines, Mr. L. Heber Cole describes the occurrences of salt in Canada and gives a very complete account of the technology of its manufacture. A chapter on the allied industries, such as caustic soda, chlorine, etc., is also included. The report contains many illustrations, maps and diagrams of salt beds, springs, and apparatus for commercial production and discusses the present status of commercial salt production in Canada.

New Portland Cement Plant.—The Fuller Engineering Company of Allentown, Pa., has just closed the contract for the new 2000-barrel-per-day Portland cement plant for the Consumers' Portland Cement Company of Chicago. This plant is to be built at Foreman, Ark., and will be modern in every respect and electrically driven throughout with individual motors. All the latest approved methods of manufacturing will be incorporated, including a complete installation of Fuller dreadnought mills.

Rail Tests.—The Titanium Alloy Manufacturing Co. has issued Rail Reports, Bulletin No. 8, describing examinations of open-hearth rails for the purpose of determining carbon segregations and merit comparison. The rails examined were Standard A and titanium-treated and the results are represented in diagrams. Sulphur prints are given of all test pieces, showing the segregation. The following summary shows the results of these segregation tests:

Segregation	Number of Pieces Standard	Titanium-treated
0-12 per cent	45	95
12-15 per cent	5	4
15-30 per cent	49	1
30-40 per cent	8	1
Over 40 per cent	4	0

A diagram representing merit percentages, calculated by the Wickhorst formula from analyses and tests made on the above samples, shows titanium treatment to have considerably increased the merit.

Flotation of Joplin-Galena Slim

BY GEORGE BELCHIC AND GLENN L. ALLEN

The Joplin-Galena slimes constitute the material that overflows from the settling tanks and deposits in slime ponds. It may be divided into two classes: (1) slimes formed from a free milling ore, and (2) slimes formed from a "chatty ore," the latter being an ore in which the minerals are finely disseminated in the gangue rock and consequently requires finer crushing. The following screen analysis will suffice to show the difference in fineness of the material, as well as the assay, of slimes formed from chatty and free-milling ores:

Table I.—Slimes formed from a chatty ore taken from a Joplin mill. This material assays 6.55 per cent metallic zinc, and 99 per cent of it passes through 150 mesh.

Table II.—Slimes made from a free milling ore, taken from a Galena, Kan., mill, and assaying 7.82 per cent metallic zinc.

TABLE I

Opening, Inches	Mesh	Sample Weight, Grams	Per Cent	Per cent Cumulative Weights	Per Cent Zinc
All pass through ...0.0082	65	....	....	....	..
Retained on..0.0058	100	0.5	0.11	0.11	2.0
Retained on..0.0041	150	3.0	0.67	0.78	3.9
Retained on..0.0029	200	87.0	19.33	20.11	5.5
Through...0.0029	200	359.5	79.89	....	6.8
Total		450.0	100.00		

TABLE II

Opening, Inches	Mesh	Sample Weight, Grams	Per Cent	Per cent Cumulative Weights	Per Cent Zinc
All pass through ...0.0110	50	....	....	....	..
Retained on..0.0091	60	9.0	1.84	1.84	3.41
Retained on..0.00675	80	5.0	1.04	2.92	2.06
Retained on..0.0055	100	12.0	2.48	5.40	2.77
Retained on..0.0036	150	21.0	4.35	9.75	3.59
Retained on..0.0020	200	268.0	55.48	65.23	7.86
Through...0.0020	200	168.0	34.81	....	10.30
Total		483.0	100.00		

Table III gives the analyses of six samples of slimes taken from various ponds in Galena, Kan., these samples covering the district geographically.

TABLE III

Sample	Per Cent Zinc	Per Cent Lead	Per Cent Iron	Per Cent Insoluble
1	3.77	0.69	4.58	85.87
2	7.82	1.09	4.46	70.75
3	5.57	0.37	10.78	78.81
4	3.69	0.41	3.08	88.87
5	7.43	0.45	7.81	81.23
6	4.20	0.26	4.18	81.27

The flotation tests were carried out with a laboratory experimental machine, and over a hundred tests were made, in order to establish the following factors:

1. Kind and amount of oil.
2. Effect of adding acid.
3. Effect of temperature.
4. Ratio of dilution of pulp.
5. Speed of agitation of propellers.
6. Time of agitation.

The oils used in the tests were pine oil, creosote oil, tar oil, crude petroleum, crude carbolic acid, cresylic acid, eucalyptus oil, rosin oil, oleic acid, rape seed oil and various combinations of the above. Tests were also made with mixtures of the above oils with carbon-bisulphide, benzine-sulphonic acid, kerosene, turpentine, etc. Summarizing the results of the numerous tests the following facts were established:

1. Good results may be obtained without the use of sulphuric acid, although a small quantity (0.5 per cent of the weight of dry slime) in many instances brought the extraction somewhat higher without, however, increasing the zinc tenor of the froth-concentrate. With Kansas crude oil the addition of sulphuric acid is detrimental to flotation. Better results may be obtained if benzine-sulphonic acid be used instead of sulphuric acid with both crude petroleum and tar oils.

2. Comparatively high temperature (60 deg. C.) gives better results with light pine oils and rape seed oil, while room temperature (25 deg. C.) is better suited for the heavy tar oils and crude petroleum.

3. Speed of agitation was found best between 1200 and 1500 r.p.m. Above 1500 r.p.m. gangue material is often thrown over with the froth. Excessive speed of rotation produces, in many instances, air bubbles as large as 2 in., which carry readily both gangue and sulphide particles.

4. Pulp dilution varies with the fineness of the slimes. With very fine slimes a 7:1 dilution was found to give the best results. With coarser slimes a 3:1 dilution worked satisfactorily. With those slimes, in which a large percentage of the particles is in colloidal form, greater dilution is necessary to suspend the particles and allow the oil to surround the sulphide grains. Mechanical adhesion of sulphide and gangue particles in the froth are readily visible in slimes of this character.

5. Time of agitation in these tests ranged from five to ten minutes. This, however, is not so important since on a commercial scale the operation may be continuous.

6. The oils that gave the best results were the following:

Pine oil (specific gravity 0.937) gave a froth-concentrate assaying 53 per cent metallic zinc, with an extraction of over 73 per cent; 2.5 lb. of oil and 10 lb. of sulphuric acid were used per ton of dry slime. The slime feed assayed 5.17 per cent metallic zinc with a trace of lead.

Rape seed oil (specific gravity 0.85) gave a froth-concentrate assaying 48 per cent metallic zinc, with an extraction a little over 55 per cent when used in quantity of 10 lb. of oil per ton of dry slime, and without the use of acid. The slime feed assayed 5.17 per cent metallic zinc with a trace of lead.

Benzine-sulphonic acid and tar oil (1:1 by weight) gave good results when used in the quantity of 8 to 10 lb. of the mixture to the ton of dry slime; a froth-concentrate assaying 56 per cent metallic zinc with an extraction of over 50 per cent was obtained. The slime feed assayed 3.69 per cent metallic zinc and 0.41 per cent metallic lead.

Carbon-bisulphide and rosin oil (1:1 by weight) gave a froth-concentrate assaying 40 per cent metallic zinc, with an extraction of 50 per cent when used in the quantity of 5 lb. of the mixture to the ton of dry slime. The feed assayed the same as the one above.

These preliminary results would indicate that the Joplin-Galena slimes, which at present are a waste product, might be treated on a commercial scale by flotation, and by this means the loss of zinc in milling in this district might be greatly reduced.

Division of State Chemical Research,
Department of Chemistry, University of Kansas,
Lawrence, Kansas.

The Driver-Harris Wire Co., Harrison, N. J., will erect a brass foundry, the contract having been let.

California Gold and Petroleum Yield.—The gold yield of 999,113 fine ounces in 1914, valued at $20,653,496, was, with the exception of 1883, the greatest of any year in California since 1864, according to the State Mining Bureau. California leads all States in this item, as well as petroleum, which is the largest single item, the 1914 figures for which were: 102,881,907 barrels, valued at $47,487,109.

Electrolytic Antimony Refining *

BY ANSON G. BETTS

It is already known that antimony may be electrolytically deposited in a very satisfactory mechanical condition from the fluoride solution, and that the deposit does not carry down antimony salt with it, as the deposit from the chloride solution is apt to do. I have shown that the fluoride solution, for an easy anode reaction, should not contain sulphates or other oxygen acids, or their equivalents, because the anode reaction tends to the formation primarily of antimony sulphate, for example, which quickly decomposes into an insoluble basic sulphate (Transact. Am. Electrochem. Soc., vol. 8, p. 187, 1905). This material insulates the anode surface, and by raising the electromotive force helps the solution of the less-readily attacked constituents of the anode. The fluorine anion is relatively sluggish, and in presence of the more easily transported sulphion, for example, forms at the anode a small proportion only of the directly soluble antimony fluoride. To achieve at once complete solution of the anode as fluoride, it is necessary to exclude all other anions from the solution.

It has not been known how the impurities of the antimony anodes will behave. Lead can be expected to form lead fluoride, which by reason of its insolubility may remain largely in the anode slime. Bismuth also makes an insoluble fluoride in the presence of free hydrofluoric acid, and in the absence of stronger acids, and may be expected to remain in the anode slime, if the solution is rightly proportioned.

Silver stands about 0.6 volt below antimony in the scale of electromotive forces of solution, and could only be expected to remain in the anode residue as metal; gold would, of course, act likewise, and for the same reason.

Copper is readily dissolved as anode in a fluoride solution, and stands so near antimony in the scale that it might seem unlikely that copper would remain undissolved if there is the usual polarization voltage at the anode surface of 0.1 to 0.2 volt, even with a pure fluoride solution. It will be shown by what follows that generally copper does not dissolve at the anode, and knowing the ready solubility of copper fluoride, we can be quite sure that the copper remains in the slime in the metallic condition.

Arsenic is also very near to antimony in the electrochemical scale, and is thought to have a higher deposition voltage than antimony. One would expect that arsenic would dissolve from the anode, and not depositing so rapidly as the antimony, would accumulate in the solution until by virtue of its increasing concentration it finally deposited in the cathode metal as fast as it dissolved from the anode. This expectation was not realized experimentally.

The first experiment was made in a paraffined wood cell holding about 1500 c.c., fitted with a stirrer which at times stopped working and detrimentally affected the experiment, but not seriously.

The anode was 1 x 4.25 x 6 in. (2.5 x 10.5 x 15 cm.) and weighed 3655 grams. Analysis showed it to contain 74.79 per cent antimony, 11.88 arsenic, 3.98 lead, 3.25 copper, 0.16 iron, 2.58 bismuth and 0.092 silver.

The solution was made up to contain in 100 c.c., 10 grams of antimony as trifluoride, besides potassium fluoride and hydrofluoric acid in the proportions given by the formula $SbF_3.KF.HF$. The mixture crystallized partly, so the solution was not as strong as was intended.

The cathodes were of copper, spaced about one inch (2.5 cm.) from the surfaces of the anode. The current used varied from two to three amperes, and mostly it was very near 2.1 amperes, corresponding to a current density of 6.6 amperes per square foot, or 73 amperes per square meter. The electromotive force was near 0.6 volt, but this increased after two or three days, as the slime layer grew thicker. The slime was removed occasionally, to prevent this undue voltage with the consequent increased solution of impurities from the anode that one would expect. The deposited metal was removed from the cathodes frequently, and the percentage of impurity it contained was determined. No appreciable quantity of impurity was found in any of it, except about 0.35 per cent of Pb, and some As. The principal results are shown graphically in Fig. 1. It was noted that the percentage of As in the metal deposited only varied a little, depending on the rate of circulation to a slight extent, but more on the current density and on the amount of slime on the anode.

The amount of arsenic in the solution at all times was small, the amount being plotted in Fig. 1. Comparatively, the As is far more concentrated in the cathode metal than in the solution, which shows it to be more readily precipitated than antimony. As/Sb in cathode metal approximated to 0.065, while As/Sb in solution approached 0.016.

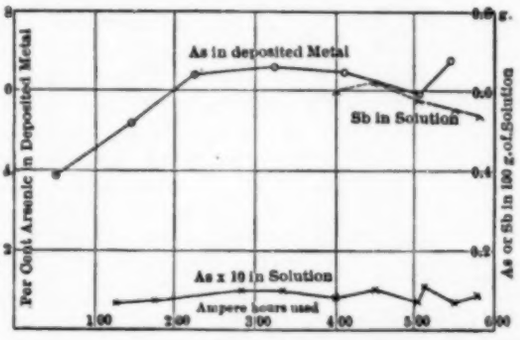

FIG. 1—AMOUNT OF ARSENIC IN SOLUTION AND IN CATHODE DEPOSIT

The deposited metal was at all times hard, solid and non-projecting.

The principal conclusions to be drawn are: (1) there is no difficulty in eliminating lead, bismuth, and copper under these conditions, which are rather severe with so low a proportion of antimony in the anode metal; (2) the solution does not become contaminated with impurities; (3) the physical condition of the deposit is all that could be desired; (4) the required current density is less than is desired and generally used for other metals, and the voltage is a good deal higher; (5) only about half the arsenic could be kept in the anode slime.

The additional analyses which were made in connection with the experiment are given in Table I:

Samples of the cathode metal for analyses were fused under KCN, before the analyses. The weight of anode decomposed was 853 grams, and of metal deposited, about 675 grams. There was 199 grams of slime produced. Some metal was found in the bottom of the cell.

In the preceding experiment the anode contained a larger percentage of impurity than is usual in electrolytic refining of metals. As it was impossible to conclude that with an anode containing less impurity arsenic would remain entirely, or in larger proportion, in the anode residue, another experiment was necessary.

Part of the old anode was melted with an equal

* A paper presented at the San Francisco meeting of the American Electrochemical Society, September, 1915.

TABLE I

	1 Per Cent	2 Per Cent	3 Per Cent	4 Per Cent	5 Per Cent	6 Per Cent	7 Per Cent
SLIME:							
Arsenic	19.6	....	18.06	17.46	....	....	....
Antimony	28.1	....	21.17	24.57	....	....	....
Lead	15.4	....	14.3	14.2	....	....	....
Copper	10.6	....	16.3	16.0	....	....	....
Bismuth	9.85	13.5	13.1	11.7	....	....	....
Iron	Trace	....	....	....	....	....	....
Silver	....	....	....	....	....	....	....
Amount	48 gr.	30 gr.	39 gr.	46 gr.	36 gr.	....	....
DEPOSITED METAL:							
Arsenic	3.89	5.18	6.36	6.59	6.45	5.92	6.81
Antimony	94.73	93.32	91.97	91.33	91.88	....	....
Lead	....	....	....	....	0.35	....	....
Copper	....	....	....	....	None	....	....
Bismuth	....	....	....	....	None	....	....

quantity of fresh antimony, and was cast into a new anode, which should have had about the following composition: 87.5 per cent antimony, 5.95 arsenic, 2.0 lead, 1.65 copper, 1.3 bismuth, and 0.16 iron. (It was not analyzed.)

The solution and cathodes were the same as were used in the former experiment. The current density was about 6 amperes per square foot, and electromotive force 0.6 to 0.65 volt. The electrolysis was continued for about 36 hr., and 250 grams of the anode was decomposed, while 221 grams was deposited on the cathodes. The anode slime was very soft and readily removed, and did not interfere with the solution of the metal underneath, as was the case in the first experiment.

The electrolyte was analyzed at intervals for As, which was found to amount to 0.06 gram As per 100 c.c. during the entire experiment. The first 143 grams deposited contained As, 4.23 per cent; Cu, Pb, Bi, none, and the 78 grams following contained 13.9 per cent As and 25.3 per cent Sb.

The conclusions are these: (1) With an anode of this composition the electrolysis goes smoothly, and there is no need of scraping the anode slime from the anodes; (2) a larger proportion of the arsenic goes into the cathodes, viz.: 60 per cent as against about 40 per cent of the total As in the first experiment. The deposit was physically good, perhaps a little better than before. Cu, Pb and Bi may be separated from antimony in this way.

When a solution of arsenious chloride is heated with metallic copper, a black arsenide of copper is precipitated on the copper. To learn whether the presence of copper would tend to retain arsenic in the anode slime as an arsenide of copper, thus yielding a purer antimony deposit at the cathode, an anode was made containing about 84.8 per cent antimony, 5.4 arsenic, 1.8 lead, 6.65 copper and 1.2 bismuth.

In refining this anode, the solution and conditions were the same as in the preceding experiment, except that the stirring was even more vigorous this time. At the end of 31 hr., the solution contained 0.045 per cent As and the deposited metal 3.15 per cent As, with only a trace of copper.

The result shows the effect of copper in holding back arsenic in the slime to be certainly small, if there is any such effect at all.

Following up the discovery that arsenic tends to deposit from the fluoride solution more rapidly than antimony, in proportion to the quantities present, it becomes important to know if in depositing antimony from the fluoride solution containing sulphuric acid, with an insoluble lead anode, as is sometimes done in treating Pb refinery anode slime, the arsenic present in the solution could not be collected with the first part of the antimony, thus producing the latter, and perhaps, greater part of the antimony in pure form.

One thousand four hundred cubic centimeters of solution was used containing about 124 grams of antimony and 5.3 grams of As as trifluorides, 1 or 2 grams HF, and 65 grams H_2SO_4; or per 100 c.c. Sb 8.85 grams, As 0.38 grams, HF about 0.1 gram, H_2SO_4 4 grams. The solution was electrolyzed at the ordinary temperature with two copper cathodes, having an active area of 24 sq. in. (150 sq. cm.), and with a lead anode, cotton-covered, having an active area of about 2.5 sq. in. (15.6 cq. cm.). The current was 1.2 amperes, or a cathode current density of 7.2 amp. per square foot (80 per sq. m.).

Analysis of the solution was made at intervals for antimony and arsenic, the difference between two consecutive analytical results showing the quantities deposited. The results are plotted as Fig. 2, in which the curves show grams of antimony, and twenty times the grams of arsenic per 100 c.c.

Only one analysis of deposited metal was made, that of the first 9 grams deposited, which contained 5.0 per

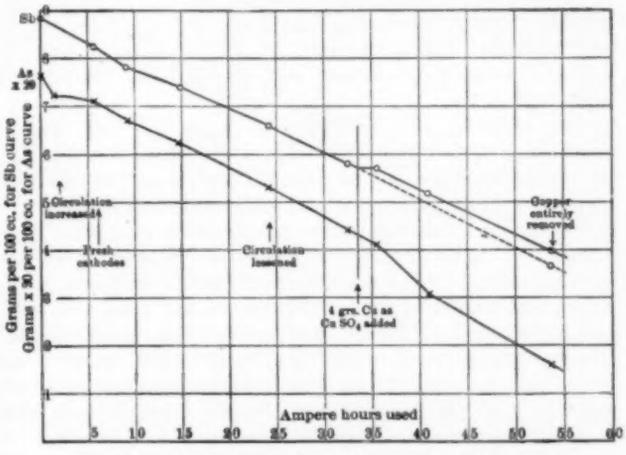
Sb
As x 20
Grams per 100 cc. for Sb curve
Grams x 20 per 100 cc. for As curve
Circulation increased
Fresh cathodes
Circulation lessened
4 grm. Cu as Cu SO4 added
Copper entirely removed
Ampere hours used
5 10 15 20 25 30 35 40 45 50 55 60

FIG. 2—ANTIMONY AND ARSENIC ANALYSES

cent As. The proportion of As to Sb at the start was 4.3 per cent.

Referring to the figure, the Sb and As curves, up to 34 amp. hr. show that the Sb and As are depositing out in very nearly the same proportion as they exist in the solution. If any separation of the two were practicable in this way, the favorable conditions of this experiment, low current density, and very good circulation, should show it. During the experiment the rate of circulation was changed to see if it made any difference, but none was detected.

After about 34 amp. hr. of current passed, and it was seen that no practicable separation of As and Sb was indicated with this solution, copper sulphate was added to the solution, to see if copper in depositing would have any tendency to carry As out with it. Furthermore, copper is usually present to some extent in the antimony fluoride solutions resulting from the treatment of lead-refining slimes by hydrofluoric acid, and in removing that copper by electrolysis with a very low density, arsenic might be found to separate with it.

The conclusion of the experiment showed that copper has a tendency to do this, but the effect is too small to be of any use.

The continuation of the Sb curve in a dotted line is derived by subtracting from the antimony found by analysis, the antimony equivalent of the copper added.

Calculating the ratios of antimony and arsenic in the deposited metals and in the solution, in the beginning of this experiment they are respectively about 19.1 and

23.1, so that the relative speed of deposition of antimony and arsenic in proportion to the amounts present, is about 1:1.2.

With the solution used in the antimony refining experiments, the principal difference being that the refining solution contained no sulphuric acid, the corresponding ratio is about 1:4.2. The presence of sulphuric acid interferes to a large extent with the tendency toward separation of the two elements.

With such a refining solution as that used in the first experiment described, there is sufficient difference in the behavior of arsenic and antimony to make possible a differentiation of the deposited metal into two parts, one a good deal lower in arsenic and the other higher, by alternately causing the refining solution to circulate rapidly and remain at rest. If the solution were electrolyzed quietly, keeping the anode away from the cathode, and removing a large proportion of the metal from the solution in contact with the cathodes, the deposited metal should contain only about 1 per cent of As instead of 6 per cent, as the total quantity of As present in the solution is so small. If the solution is then rapidly circulated so that the As brought into solution from the anode could be quickly brought to the cathodes, the proportion of As deposited should be much greater than 6 per cent, perhaps as high as 25 per cent.

The simplest way in which to carry out this idea of producing two quantities of antimony, one very high and one very low in arsenic, is to cause the solution to pass in turn through a cell with strong agitation, and one with as little agitation as possible, preferably arranged so that the heavy solution running down the anode surface could be collected in the bottom of the quiet cell, and siphoned out into the agitated cell.

The apparatus used consisted of one main wooden cell 38 cm. deep by 12.75 cm. by 20.3 cm., with a capacity of about 10 liters, and a series of four auxiliary cells 12.9 cm. deep by 7.75 cm. by 7.75 cm., with a total capacity of about 3 liters.

The main anode, about 1.8 cm. thick by 15 cm. by 30 cm. was suspended in the middle of the main tank, between two copper cathodes.

The first of the small cells, through which the solution from the main tank passed, contained one anode of the same composition as the main anode, and a copper cathode, the active anode or cathode area being about 6.8 cm. by 10 cm.

In the other three tanks anodes of pure Sb, and Cu cathodes were used, of the same size as those in the first of the small tanks. The alloy to be refined contained 84.0 per cent antimony, 5.4 arsenic, 3.6 copper, 3.7 lead and 1.1 bismuth.

The solution was made to conform to the formula $SbF_3.NaF.KF.HF$, and should have contained 4.4 grams Sb, 1 gram Na, 1.4 grams K, and 0.8 gram HF per 100 c.c. A small quantity of salts crystallized out and a very little solution was spilled, so the solution was undoubtedly somewhat weaker and contained at the end of the experiments only 1.8 per cent Sb. This lowering would, of course, result from the conversion of the lead in the anode to lead fluoride with the reduction of the corresponding amount of antimony.

Some difficulty was experienced in regulating the flow of solution through the tanks, and the stirrers sometimes stopped during the night.

With an electromotive force of 1.35 volts the large tank used (at about 17 deg. C.) 8.5 amp., and the four small tanks in multiple about 15 amp., or current densities of respectively 1.2 and 0.4 amp. per square decimeter (12 and 4 per square foot).

The large tank was fitted with a distributer, so that the heavy solution flowing in from the four small purifying tanks should mix with the light solution at the top of the large refining cell.

The heavy solution running down the anode collected in the bottom of the tank and was continuously removed to the purifying tanks through a rubber tube.

During the first experiment the heavy solution at the bottom of the refining cell and in the purifying cells contained about 6.2 per cent Sb, and the solution at the top of the refining cell 3.3 per cent Sb.

The character of the deposit in the four purifying cells was different, that in the first cell especially being darker, duller and less crystalline, and containing more As, as was expected.

After three days the solution in the first three of the purifying cells contained:

No. 1	0.148 per cent arsenic
No. 2	0.133 per cent arsenic
No. 3	0.122 per cent arsenic

The purification was evidently less complete than could be inferred from the fact that the solution in the first refining experiment contained only about 0.05 per cent As.

After five days the upper three-quarters of the main cathode was collected and melted separately from the bottom quarter. The upper 660 grams contained 3.0 per cent As and the lower 210 grams contained 3.4 per cent As. To the solution was then added enough NaF to correspond to the formula $SbF_3.2NaF.KH.HF$. The solution crystallized a little, so it was taken out, filtered, mixed and analyzed, showing it to contain 0.07 per cent As, and put back.

After five days more 1060 grams of cathode metal was found to contain 3.18 per cent As.

The analyses of Table II are of the slime produced in the first five days, the second five days, and that taken from the first small cell.

TABLE II

	First Slime, per Cent	Second Slime, per Cent	Slime in Small Cell, per Cent
Arsenic	8.2	8.17	8.5
Antimony	7.16	7.94	16.7
Lead	26.9	28.1	
Copper	30.1	26.2	
Bismuth	6.53	5.4	

Thinking that a diaphragm might keep the impure anode solution in the refining cell away from the cathodes, a muslin diaphragm was used, reaching nearly to the bottom of the cell. The diaphragm was of two or three thicknesses, spaced ½ in. (1.25 cm.) in the clear from the anode surface. The current was reduced to 5.5 amp. for the large cell and 1.5 amp. for the four small cells in multiple. After six days electrolysis the figures of Table III were obtained by analysis of the products:

TABLE III

	Grams	Per Cent Arsenic
Upper third	240	3.4
Middle third	275	3.3
Lower third	260	3.3
Small tank No. 1	59	4.6
Small tank No. 2	87	3.9
Small tank No. 3	75	3.9
Small tank No. 4	100	3.9

The absence of a diaphragm was evidently not the weak point in the first part of the experiment, as the result was practically the same either way. The percentage of As in the metal deposited in the four small tanks was evidently much less than indicated from the

ratio of 1: 4.2 for As in metal found in the first experiment. The difference must be due to a difference in the temperature, current density, or composition of the solution.

Another run of four days with the addition of four grams HF per liter yielded products analyzing as stated in Table IV.

TABLE IV

	Grams	Per Cent Arsenic
Main cathode metal	430	3.6
Small tank No. 1	17	6.6
Small tank No. 2	23	6.3
Small tank No. 3	20	6.1
Small tank No. 4	20	5.8
Total of small tanks	80	6.05

The improved, but still unsatisfactroy result, is believed to be due to the presence of more HF in the solution.

The solution (now thrown away) contained 1.8 per cent Sb and 0.06 per cent As, giving a ratio calculated as before of 1:2 since

$$\frac{\text{As in solution}}{\text{Sb in solution}} = 0.033 \text{ and } \frac{\text{As in metal}}{\text{Sb in metal}} = 0.066.$$ Ratio is 1:2

It was concluded that a study of the ratio

$$\frac{\text{As in solution}}{\text{Sb in solution}} : \frac{\text{As in metal}}{\text{Sb in metal}}$$

with varying solution composition, temperature, current density, and perhaps other factors is necessary for successful refining by this method.

Experiments were then made to determine the effect of temperature on the amount of As passing over into the cathodes. As there is a counter e.m.f. even with the fluoride solution, sometimes as high as 0.2 volt, it might be possible by raising the temperature to reduce the anode e.m.f. to a point where antimony but not arsenic could be dissolved.

The solution used contained, in 100 c.c., 3 grams NaF, 11 grams SbF_3 and 1 gram HF. Three small rubber jars were arranged as electrolytic cells at temperatures of 30 deg., 40 deg., and 50 deg., respectively. The current varied from 1 to 1.4 amp., corresponding to a current density per square foot of 14 to 20 amp., with an e.m.f. per cell of about 0.9 volt. After 43 hr. electrolysis the experiment was discontinued with these results:

Thirty degree cell short circuited, 35 grams deposited, in smooth condition at the bottom and rough and flaky on top.

Forty degree cell, 54 grams deposited, rather smooth and even throughout.

Fifty degree cell, 45 grams deposited, in still smoother condition.

The deposits were melted and analyzed for As, with these results:

Temperature, Degrees	Arsenic, per Cent
30	3.5
40	3.6
50	3.9

I tried the purification of antimony from arsenic by crystallizing the melted metal analogously to the purification of lead and bismuth by crystallization. Numerous experiments with antimony-arsenic alloys containing from 3 to 8 per cent of arsenic gave in all cases crystals and liquid portions containing the same percentages of arsenic.

The alkaline sulphide solution of antimony was tried as a refining medium. The solution was made by dissolving 30 grams of SbF_3 in 500 c.c. sodium sulphide solution of specific gravity 1.19. The solution was electrolyzed warm with the same composition of anode as used in the last preceding refining experiments.

After 24 hours' run the cathode deposit averaged 1.0 per cent As and after 48 hours 2.1 per cent As.

Returning to the original idea of refining in the usual way, with the fluoride solution, some of the refined antimony containing about 3 to 3.5 per cent arsenic was cast in the anode and again refined in the same way. About half of the arsenic remained in the anode slime. By several refinings the arsenic may probably be eliminated.

The most positive reaction noted in these experiments is that due to the composition of the solution. An increase in HF is highly beneficial and the presence of sulphates or sulphuric acid is shown to be objectionable. It is likely that the hydrofluoric acid may have contained traces of sulphuric or fluosilicic acid, and the negative results might have been traced to that cause. The effect of alkaline fluorides and ammounium fluoride should be studied, also.

A successful method of electrolytic refining of antimony should receive technical application, inasmuch as present methods do not save the gold frequently present and are incapable of making a product of a high and uniform grade at all comparable to the copper, lead and silver on the market.

Gas Analysis

Technical paper 104 of the Bureau of Mines deals with the "Analysis of Natural Gas and Illuminating Gas by Fractional Distillation at Low Temperatures and Pressures," the authors being G. A. BURRELL, F. M. SEIBERT and I. W. ROBERTSON.

"This publication, which is one of a series dealing with the increase of efficiency in the production and utilization of fuels, describes in detail the results of experiments, made by the Bureau of Mines, with a method of separating and determining the hydrocarbons in gaseous fuels. The method employs fractional distillation in a vacuum at low temperatures. The gas is liquefied, the different constituents separated by proper adjustment of temperatures, the various fractions removed with a mercury pump, and these fractions analyzed by ordinary slow-combustion methods. The method was successfully applied to the separation of hydrocarbons in natural gas and in artificial illuminating gas, and is the only known method applicable to the separation of some hydrocarbons."

The first part of this report describes the results of experiments by which the natural gas used in Pittsburgh, Pa., was separated into its individual paraffin hydrocarbons. The second part describes experiments in which the separation of the illuminants in artificial illuminating gas, that used in Pittsburgh, Pa., and that used in New York City, was effected. Some experiments to determine the practicability of separating gases whose boiling points lie close together, such as ethane and ethylene, are included.

"Each year much experimental work is done in the destructive distillation of different fuels, liquid, gaseous and solid, which yield complex mixtures of different hydrocarbon gases. In reports on these tests some of the gases are not separated, but are invariably lumped together as illuminants (determined by absorption in fuming sulphuric acid, or bromine water) or calculated in the combustion analysis as methane and ethane. By the methods described in this publication the gases can be separated into the individual constituents. Hence a wide field of investigation is opened up, in that a much more exact study of combustion processes can be made and chemical changes more closely followed and better understood."

The Electrolytic Precipitation of Gold, Silver and Copper from Cyanide Solutions*

BY G. H. CLEVENGER

(*Concluded from page* 806)

Practice at San Sebastian, Salvador

The original electrolytic precipitation plant installed at Butters' Salvador Mines, San Sebastian, was similar to the Minas Prietas plant.

However, after being placed in operation it was found necessary to modify the practice in several important particulars. The two wooden containing boxes used were 30 ft. (9 m.) long, 10 ft. (3 m.) wide, and 4 ft. 8 in. (1.4 m.) in depth. The bottom was given a slope to one side at the rate of 1 in. per foot (8 cm. per m.). Each box was divided into twelve compartments, ten of which were used for precipitation, the last two merely serving as settlers. Each compartment contained 25 anodes and 24 cathodes.

The anodes were rolled lead plates ⅛ in. (0.3 cm.) thick, 22 in. (55 cm.) wide and 48 in. (120 cm.) long. For some time the anodes were peroxidized in a special cell using an electrolyte consisting of 1 per cent of potassium permanganate and 2 per cent sulphuric acid; but later untreated lead plates were used.

At first tinplate cathodes similar to those used at Minas Prietas were tried, but certain difficulties arose which made it necessary to abandon the tinplate cathode. The original intention was to deposit the gold and copper as a non-adherent slime, as at Minas Prietas, but despite the fact that a higher current density was used the gold and copper were deposited upon the cathodes as a quite firmly adherent coating.

The predominating metal in the solution at Minas Prietas was silver, but at San Sebastian it was copper. The well-known fact that silver is very prone to give non-coherent coatings, while copper has a strong tendency to give adherent coatings, seems to offer a ready explanation of this difference.

The removal of the strongly adherent copper-gold plating from the tin cathodes presented a most difficult problem. Various expedients for removing it were tried, such as heating the cathodes with a gasoline torch, scraping and pounding them, etc. After these various heroic measures had proved ineffective, it was decided to return to the lead-foil cathode cut into strips, as used in the Siemens-Halske process. When it was desired to "clean up," the lead-foil strips were removed, ground and shipped to smelters for the separation of the metals.

The current was supplied by a belt-driven Siemens-Halske dynamo giving 450 amp. at 30 to 35 volts. The current density was 10 amp. per square foot (110 per sq. m.) of anode surface, and the voltage across anodes and cathodes was 4 to 4½.

This general practice was continued until C. P. Richmond modified it so that the metals could be separated locally. The lead-foil cathodes were replaced by sheet-lead cathodes 1/16 in. (0.15 cm.) thick, 22 in. (55 cm.) wide and 48 in. (120 cm.) long, so arranged that they could be readily removed from the containing box. To this end, the connecting wires for the cathodes, instead of being soldered to the copper bus bars, were clamped by wing-nuts in split studs screwed into the bus bars.

The general arrangement of the preciptation box is shown in Fig. 3. The wires from five anodes and cathodes of opposite compartments were laid in each

*A paper read at the San Francisco meeting of the American Electrochemical Society, September, 1915.

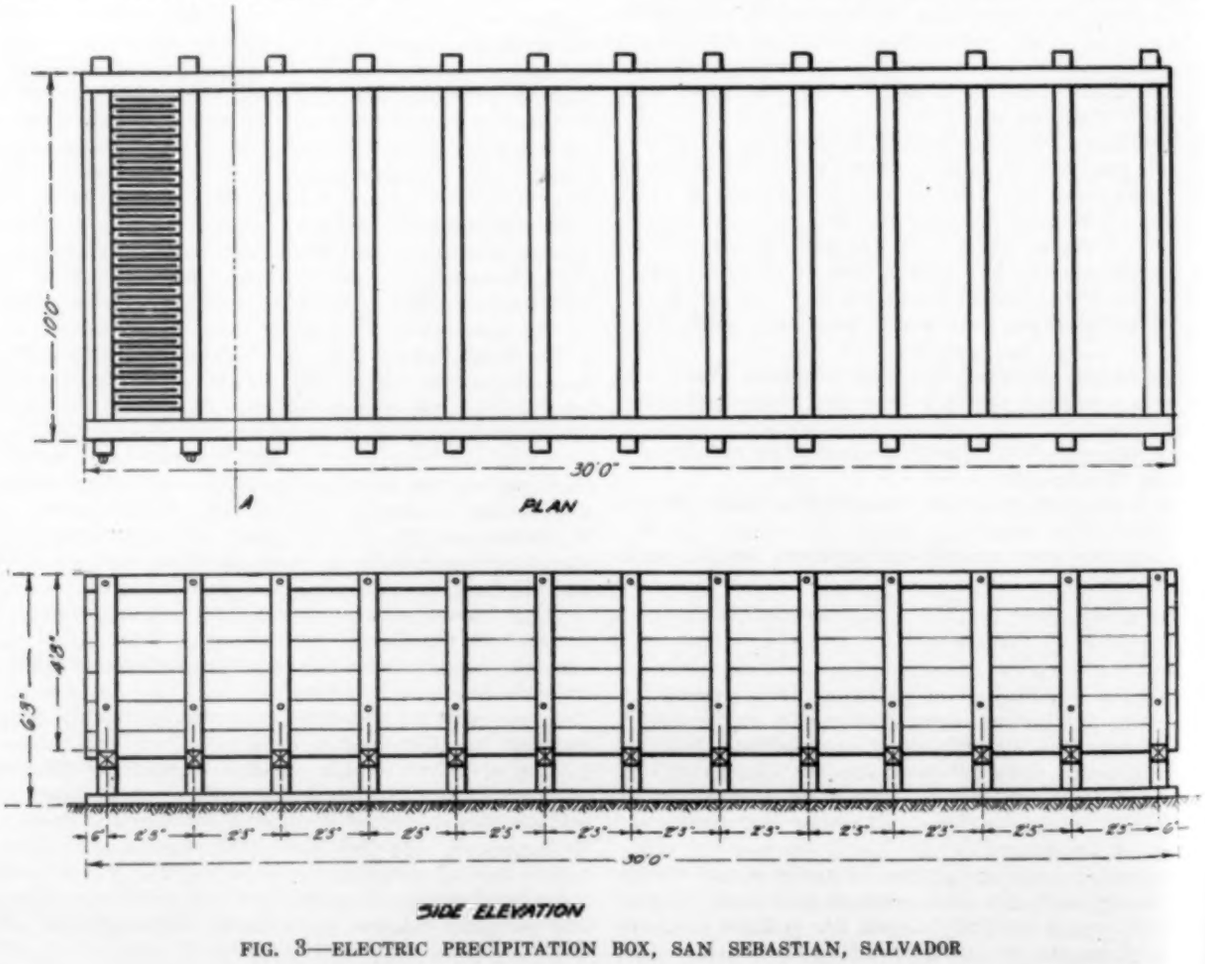

FIG. 3—ELECTRIC PRECIPITATION BOX, SAN SEBASTIAN, SALVADOR

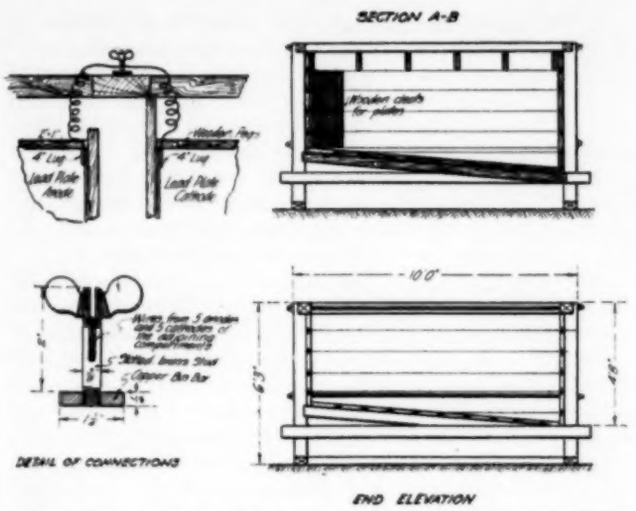

FIG. 3A—DETAILS OF FIG. 3 (ELECTRIC PRECIPITATION BOX)

slot and a tight connection secured by screwing down the wing-nut. The studs were spaced 19 in. apart and six of them connected the cathodes of one compartment to the anodes of the adjoining compartment. By this arrangement it was possible to remove any of the cathodes without interrupting the flow of current or solution.

At San Sebastian it was found advantageous to operate so as to keep the concentration of the copper down to a point where it did not interfere with gold extraction, and precipitate the remaining gold by zinc shaving. This was done despite the fact that zinc was said to interfere with electrolytic precipitation in South Africa.

The solution leaving the electrolytic boxes was divided between two zinc boxes, each 22 ft. (6.6 m.) in length and having 14 compartments, 2 ft. (60 cm.) by 1 ft. (30 cm.) and 2 ft. (60 cm.) deep, filled with zinc shaving. The ore treated was a high-grade gold ore containing considerable pyrite and sufficient copper left in the soluble form after the roasting which preceded cyanide treatment, to occasion trouble in cyanidation unless electrolytic precipitation were used.

The average gold content varied, but at the time that electrolytic precipitation was in use ranged from 2 to 4 oz. (58 to 117 g.) per ton. The soluble copper varied from ½ to 2½ lb. (0.23 to 1.14 kg.) per ton. The ratio of gold to copper in the electrolytic deposit ranged from 1 to 7 up to as high as 1 to 25, but generally was between the limits of 1 to 9 and 1 to 14.

The average results of two months, taken at random, will give a general idea of the performance. During this period there was treated 45 tons of ore daily, averaging 2.27 oz. (66.2 g.) gold per ton. Approximately 150 tons of solution was precipitated daily. The average content of the head solution was: Gold, 0.79 oz. (13.0 g.) per ton; copper, 0.075 per cent; cyanide, 0.118 per cent. The average content of the tail solution from the foot of the zinc box was: Gold, 0.004 oz. (0.117 g.) per ton; copper, 0.048 per cent; cyanide, 0.154 per cent. The percentage of gold in the deposit at the head of the electrolytic box was about 7; at the foot of the box it was about 3. The current density was 1.0 amp. per square foot (11 per sq. m.) of cathode surface. The voltage across the anodes and cathodes of each compartment was 4 to 4½.

During the treatment of the ore there was consumed 7.2 lb. (3.27 kg.) of sodium cyanide per ton. There was precipitated in the electrolytic boxes 1.5 lb. (0.68 kg.) of copper and apparently there was regenerated 1.92 lb. (0.87 kg.) of cyanide, in terms of potassium cyanide, per ton of ore treated. Eighty to 90 per cent of the gold and most of the copper was precipitated in the electrolytic boxes. Passage through the zinc boxes brought the total percentage of gold precipitated up to 99.5.

The zinc boxes precipitated little copper, despite the fact that the solution invariably contained copper at this point. Under normal conditions the percentage of copper in solution remained more or less constant, any increase through contact with the ore being precipitated by the electrolytic boxes. When the copper built up in the solution, as was sometimes the case through treating ore containing an unusually high percentage of copper, the bad effect was at once noted upon the gold extraction.

ELECTROLYTIC REFINING OF THE CATHODE DEPOSIT

After the cathodes had remained in the precipitation boxes 20 to 30 days, during which time they had increased in weight from 8 to 12 lb. (3.5 to 5.5 kg.), they were removed and, after draining, were connected as anodes in a special electrolyzing cell shown in detail in Fig. 4, containing an electrolyte of 2 to 3 per cent sulphuric acid, which was thrown away when its strength was reduced to 1 per cent.

They were here enclosed in compartments so constructed as not to interfere with the free passage of the current, and which served to collect the gold slime freed by solution of the copper of the deposit. The compartments consisted of a wooden frame having a pocket at the bottom somewhat larger than the cathode plate, covered on both open sides with cotton twill held in place by tacking. The cathodes used in the refining boxes were lead plates 1/16 in. (0.16 cm.) thick, 22 in.

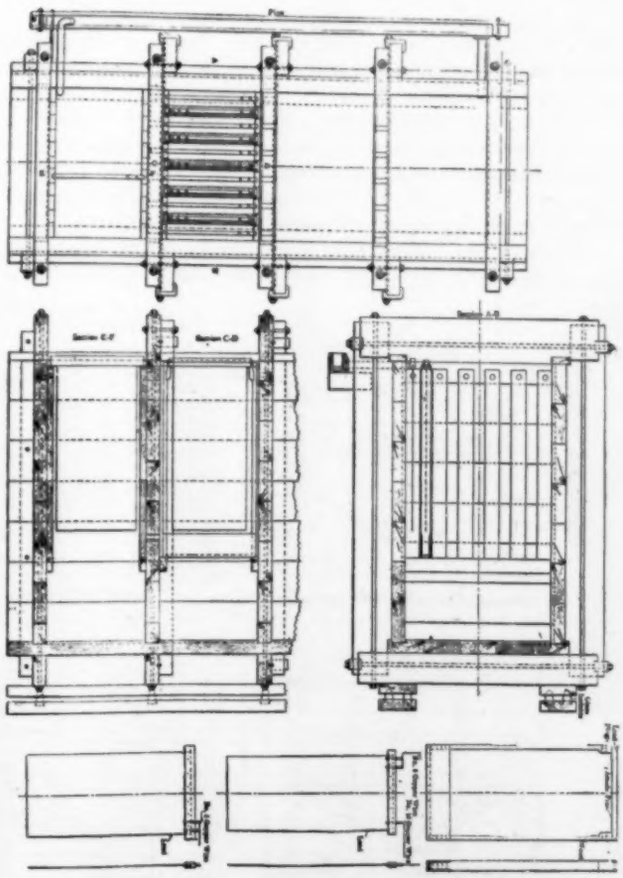

FIG. 4—ACID REFINING TANK, WITH DETAILS OF ANODE AND CATHODE

(55 cm.) wide and 48 in. (120 cm.) long. The refining box contained four compartments, each holding 5 anodes and 6 cathodes. The spacing between the plates was 4 in. (10 cm.). The compartments were connected in series and received a total current of 450 amp.

The average voltage across the electrodes of each compartment was 8. The current density was approximately 5 amp. per square foot (55 per sq. m.) of anode surface.

The copper was deposited at the cathode as a flocculent non-adherent precipitate, which collected in the bottom of the main cell, while the gold slime remained inside of the cloth-sided compartments surrounding the anodes. When the operation had proceeded to the point where all of the copper of the copper-gold deposit was dissolved, the anode plates were removed, washed and replaced again in the regular precipitation box as cathodes. The time required for dissolving the deposit on a plate was from 48 to 72 hr.

Two electrical connections were provided for each cathode; one, a No. 10 copper wire, used when the plate functions as cathode in the regular precipitation box, and the other a No. 6 wire, which is used for carrying the heavier current employed in the refining cell. Both wires are permanently soldered to the lead lugs cast on the plate.

Beginning at the head of the precipitation box the cathodes were removed and, as refined, replaced, thus gradually working down toward the foot of the precipitation box. By the time that the foot of the box was reached, the cathodes at the head were again ready for refining. The total time of "cleaning up" a precipitation box was from 25 to 30 days. Consequently the operations of cleaning up and refining were going on almost continually.

The solution in the acid refining box was circulated by a lead air-lift. The interval between "clean-ups" of the refining box varied from 15 to 25 days, depending upon the quantity of copper which accumulated at the bottom of the cell.

The copper capacity bore a definite ratio to that of the anode compartments, so that when the former was full the latter was about three-quarters filled. By this time the twill sides of the receptacle were so weakened by the acid that they required replacement.

When it was desired to "clean up" the refining box, the acid electrolyte was siphoned into a storage tank and the anode receptacles were raised and allowed to drain over the refining box. They were then placed in an upright position upon a washing board supported above a tank provided with a filter bottom, the twill was cut down the sides, removed, and the gold slime washed into the tank. The wooden frames were well scrubbed and the twill washed over the same tank, after which it was burned in a dryer. When thoroughly drained, the gold slime was removed from the tank, dried and united with the ashes from the twill.

After mixture with the proper fluxes it was melted in graphite crucibles heated in an oil-fired furnace. The buttons resulting from the preliminary melts were combined, remelted and cast into bars 700 to 800 fine for shipment.

The copper slime was flushed through 1½ in. (4 cm.) holes bored in the side of each compartment 1 in. (2.5 cm.) above the bottom into a cement launder in the floor which conveyed the slime to a tank outside the building provided with a filter bottom. After draining and drying it was sampled and boxed for shipment. The dried copper precipitate contained 84 to 87 per cent metallic copper and 1.82 to 3.72 oz. (53 to 108.5 g) gold per ton. The total time required for a clean-up was 7 hr. A number of years ago, when this system of refining was in operation, the net returns from the copper more than paid all the expense of refining, melting the gold and shipping the copper.

Practice at Virginia City, Nevada

In the installation of the electrolytic precipitation plant at Virginia City, Nevada, Messrs. Charles Butters & Co., Ltd., had the benefit of the experience gained at the plants previously installed on the Rand, South Africa, at Minas Prietas, Mexico, and at San Sebastian, Salvador. This resulted in several quite important improvements.

Originally there were nine boxes, similar to that shown in detail in Fig. 5, the plan being to use three of these boxes for precipitating copper from a preliminary wash by sulphurous acid, which, when the plant first began operations, was intended to precede cyanide treatment. The other six boxes were to be used for precipitating gold and silver, together with any remaining copper from the cyanide solutions.

The preliminary treatment with sulphurous acid was finally abandoned and direct cyanidation practiced.

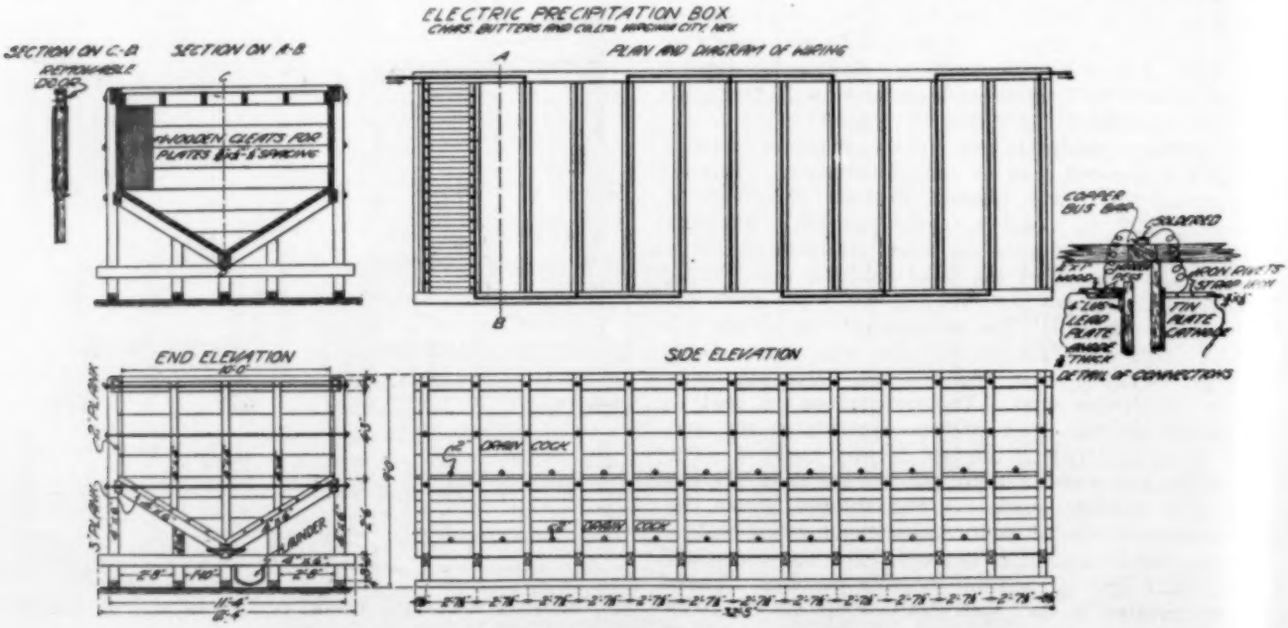

FIG. 5—PRACTICE AT VIRGINIA CITY, NEVADA

The precipitation practice finally evolved for the cyanide solution was to use four of the boxes in pairs of two arranged in tandem, the solution issuing from the first set of boxes passing through the second set. As at San Sebastian, electrolytic precipitation was followed by zinc shaving. The zinc box was improvised by placing screens in the bottoms of a number of the compartments of two of the discarded electrolytic boxes.

At Minas Prietas the method of handling the precipitate was rather crude, as a part of the cathodes and anodes had to be removed to give access to the bottom of the boxes during the "clean-up." At Virginia City this feature of the process was greatly improved by constructing the precipitating boxes with hopper bottoms. In each compartment where one of the sloping sides of the hopper joined the vertical side of the box was located a valve through which the bulk of the solution contained in the box could be drawn prior to sluicing out the precipitate. At the apex of the hopper in each compartment was located another valve through which the slime could be sluiced to a launder which conveyed it to a small concrete tank equipped with a mechanical agitator, whence it was pumped by a centrifugal pump into an ordinary distance-frame and plate type of filter press. After drying, the precipitate was sampled and shipped to the Selby Smelting Company. Later a refinery was built.

The precipitate, after retorting to recover mercury, and after the addition of the proper admixture of fluxes, was melted in graphite crucibles heated in specially designed tilting furnaces.

With the arrangement described above it was not necessary to disturb the anodes or cathodes when cleaning up.

Each box had twelve large compartments, in which were suspended 18 peroxidized sheet-lead anodes ⅛ in. (0.32 cm.) thick, 25 in. (63 cm.) wide and 48 in. (120 cm.) long, and 18 tin-plate cathodes of the same size. Sheet aluminum cathodes were also tried.

Later, an anode composed of lead rods, detail drawing of which is shown in Fig. 6, was used. The advantage of this anode was that it could be made locally from scrap lead or old anodes. The Wood dynamos, each giving 500 amp. at 50 volts, and direct connected to an induction motor, furnished the current.

The character of the ore treated at this plant varied greatly. Originally it was built to treat a large accumulation of tailing which had resulted from the treatment of Comstock ores by pan-amalgamation. Later Comstock ore and concentrator tailing, and ore from the Tonopah district of Nevada, were treated. The treatment of outside ore began before the exhaustion of the tailing, so that at times there was a varying mixture of tailing and ore treated.

The pan-amalgamation tailing had been re-treated a number of times by the same process and was, therefore, very low grade. It contained copper, mercury and chlorides as a result of the chemicals added during pan-amalgamation.

The tonnage and metal content of the solution varied considerably, but the following is an average of the results obtained over a considerable period when treating pan-amalgamation tailing exclusively, and precipitating 474 to 668 tons of solution daily. There was no distinction made between the solution originating from the sand and slime treatment, the two being combined for precipitation.

The average content of the head solution per ton was: Gold, 0.033 oz. (0.97 g.); silver, 0.78 oz. (22.7 g.); no record of copper determinations; cyanide, 0.046 per cent; protective alkali, 6 points. The tail solution from the electrolytic box contained: Gold, 0.01 oz. (0.29 g.); silver, 0.22 oz. (6.4 g.); cyanide, 0.066 per cent; protective alkali, 4 points. The zinc-box tails contained: Gold, 0.003 oz. (0.09 g.); silver, 0.03 oz. (0.88 g.).

The current density was 0.245 amp. per square foot (2.7 per sq. m.) of anode surface. The voltage across the anodes and cathodes in each compartment was 3.1.

The average content of the head solution when treating a mixture of Tonopah ore and pan-amalgamation tailing was: Gold, 0.055 oz. (1.61 g.); silver, 3.15 oz. (91.9 g.); cyanide, 0.18 per cent; protective alkali, 10 points. The tail solution from the electrolytic boxes, when precipitating 432 tons per 24 hr., contained: Gold, 0.02 oz. (0.58 g.); silver, 0.60 oz. (17.5 g.); cyanide, 0.18 per cent; protective alkali, 10 points.

The current density was 0.245 amp. per square foot (2.7 per sq. m.) of anode surface. The voltage across the anodes and cathodes in each compartment was between 3.1 and 3.2. Four lots of precipitate contained

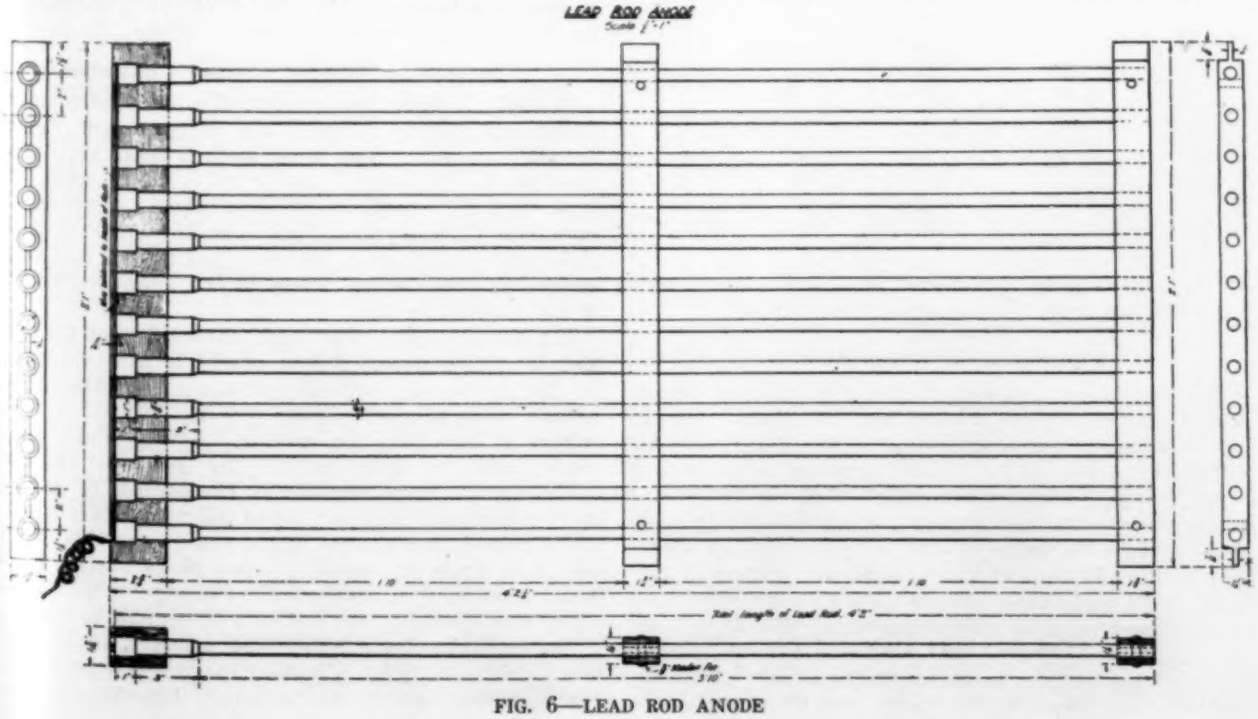

FIG. 6—LEAD ROD ANODE

the percentages of gold, silver and copper, given in Table I.

TABLE I

	Per Cent		
	Gold	Silver	Copper
A	0.23	9.60	26.60
B	0.23	8.49	29.25
C	0.29	9.99	31.40
D	0.27	7.97	20.10

Table II shows the results obtained in one of the boxes when the conditions of tonnage precipitated, current density, voltage, etc., were the same as those given above.

TABLE II
(Multiply figures by 29.17 to get grams per metric ton)

	Assay of Solution		Amount Precipitated	
	Silver, Oz.	Gold, Oz.	Silver, Oz.	Gold, Oz.
Solution entering electrolytic box	2.164	0.0403	.....	.....
Solution leaving:				
1st compartment	2.049	0.0405	0.115	0.0
2d compartment	1.756	0.0335	0.293	0.068
3d compartment	1.751	0.0335	0.005	0.0
4th compartment	1.669	0.0310	0.082	0.0025
5th compartment	1.418	0.0265	0.251	0.0045
6th compartment	1.427	0.0275	0.0	0.0
7th compartment	1.271	0.0240	0.156	0.0035
8th compartment	1.193	0.0215	0.078	0.0025
9th compartment	1.124	0.0205	0.069	0.0010
10th compartment	1.022	0.0175	0.102	0.0030
11th compartment	0.886	0.0140	0.136	0.0035
12th compartment	0.824	0.0105	0.062	0.0035
Solution leaving:				
1st zinc	0.417	0.0075	0.407	0.0030
2d zinc	0.225	0.0045	0.192	0.0030
3d zinc	0.048	0.0015	0.177	0.0030

Table III shows the composition in gold and silver deposit, as determined by corrected assay of the cathode deposits in the various compartments of the precipitation equipment when treating exclusively Tonopah ore.

TABLE III

	Percentage, Silver	Percentage, Gold	Percentage Base Metals and Other Impurities, by Difference
1st electrolytic compartment	87.69	1.98	10.33
2d electrolytic compartment	88.38	1.98	9.64
3d electrolytic compartment	80.02	1.82	18.16
4th electrolytic compartment	81.43	1.89	16.68
5th electrolytic compartment	77.95	1.77	20.28
6th electrolytic compartment	77.50	1.81	20.69
7th electrolytic compartment	67.09	1.48	31.43
8th electrolytic compartment	69.71	1.58	28.71
9th electrolytic compartment	58.33	1.37	40.30
10th electrolytic compartment	62.39	1.41	36.20
11th electrolytic compartment	55.72	1.25	43.03
12th electrolytic compartment	49.96	1.12	48.92
1st zinc compartment	59.38	1.37	39.25
2d zinc compartment	46.20	1.03	52.77

In the case of the electrolytic box, each sample was taken directly from the surface of the cathode.

Practice at El Rayo, Mexico

The equipment installed at El Rayo followed rather closely the lines of that used at San Sebastian, Salvador, and consisted of one wooden box 10 ft. (3 m.) wide by 20 ft. (6 m.) long, 4 ft. (1.2 m.) deep on one side and 5 ft. (1.5 m.) deep on the other, with the usual arrangement of wide and narrow compartments. In each of the seven wide compartments were installed 25 sheet-lead anodes, 20 in. (50 cm.) by 40 in. (100 cm.) and ⅛ in. (0.37 cm.) thick. Alternating with the sheet-lead anodes, there were 24 sheet-copper cathodes, 20 in. (50 cm.) by 40 in. (100 cm.) and 1/32 in. (0.16 cm.) thick. Electrical connection of the anodes and cathodes to the bus bars was made by means of lead plugs cast onto the plates, to which were soldered copper connecting wires. The current was supplied by a Crocker-Wheeler generator giving 500 amp. to 50 volts.

At the time that this installation was made zinc-dust precipitation of the gold and silver was giving satisfactory results. A change in the character of the ore indicated that trouble might arise through a greatly increased proportion of copper. It was, therefore, planned to continue to precipitate the gold and silver by zinc and depend upon electrolytic precipitation to keep the copper in solution down to a concentration which would not interfere with cyanide treatment. To this end 200 tons of solution, the total flow from the zinc precipitation equipment, was passed through the one electrolytic unit. The intention was to deposit the copper in as pure a state as possible so that it could be marketed directly. Certain difficulties arose in accomplishing this end, and before they were overcome, the proportion of copper in the ore had decreased to a point where it was no longer necessary to use electrolytic precipitation.

The equipment was not dismantled and stood as originally installed until the exigencies of the Mexican revolution caused the management to turn to it in an effort to conserve both cyanide and zinc. In this case the original practice was reversed. The cyanide solution was passed directly from the treatment tanks to the electrolytic precipitation box and from thence to the zinc-dust precipitation equipment. There was an average of 200 tons per day of solution precipitated. The average content of this solution was as stated in Table IV.

TABLE IV

Simple alkaline cyanide	0.15 per cent
Total cyanide	0.38 per cent
Alkalinity	100 points
Copper	0.12 per cent
Zinc	0.05 per cent
Gold	0.30 oz. (8.8 g.) per ton
Silver	5.00 oz. (146 g.) per ton

Approximately 65 per cent of the gold and silver, 20 per cent of the copper, and a small amount of zinc were precipitated by electrolysis. There was a gain in cyanide strength of about 0.075 per cent, with a considerable decrease in the alkalinity. The current density used was 1.2 amp. per square foot (13.2 per sq. m.) of anode surface. Under these conditions of operation the lead anodes disintegrated very rapidly.

Practice at Nelson, British Columbia

In 1907, Douglas Lay described a modified Siemens-Halske installation in use at the Reliance mill, Nelson, British Columbia. The three wooden containing tanks were 35 ft. (10.5 m.) long, 5 ft. (1.5 m.) wide, and 3 ft. (0.9 m.) deep, inside dimensions. The anodes were No. 10 sheet iron (0.35 cm. thick), 5 ft. 1 in. (153 cm.) by 2 ft. 8 in. (80 cm.), set in saw cuts three inches (7.5 cm.) apart, made in the sides and bottom of the containing tank.

The alternate downward and upward flow of the solution, by which means it was caused to pass uniformly by the anode and cathode surfaces, was attained by setting each alternate anode 1½ in. (4 cm.) off the bottom of the box, the solution passing under these anodes and over those set into the bottom of the box.

The anodes were enclosed in cheesecloth to prevent short circuiting. Connections to the main positive wire, which rested in a groove in the top of one side of the box, was made by means of lead plugs cast in holes bored downward to intersect the top edge of the anodes where they rested in the saw cuts in the side of the box.

The cathodes were sheet lead about 1/40 in. (0.6 mm.) thick, made by a special casting process at the Trail smelter, and of approximately the same area as

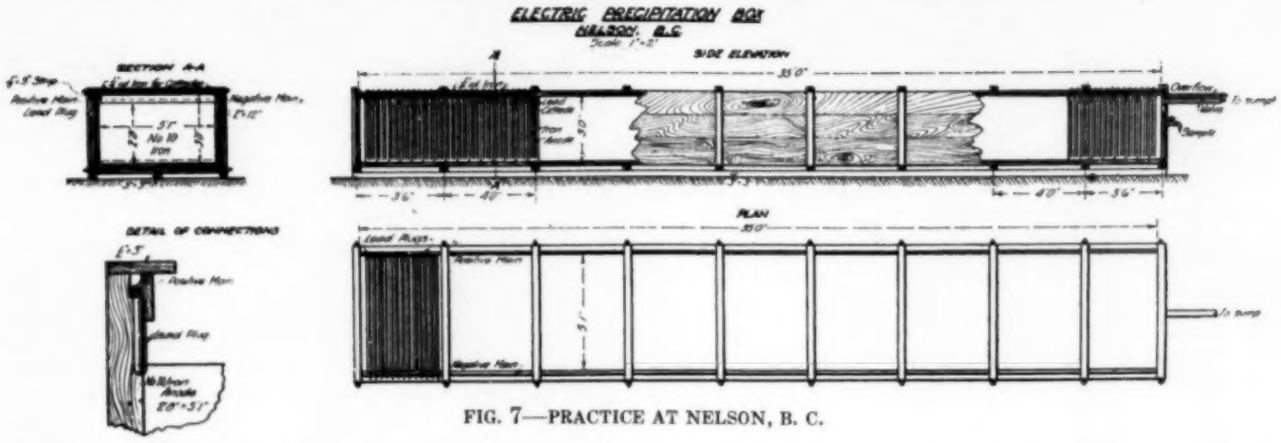

FIG. 7—PRACTICE AT NELSON, B. C.

the anodes. For convenience in handling, each cathode was made up of three sheets, each weighing about 7 lb. (3 kg.), attached to a ¼-in. (0.6 cm.) round iron cross-bar by two strips of roofing tin.

The cathodes previous to being used were dipped momentarily in a slightly acidic solution of zinc chloride to clean them thoroughly. Connection of the cathodes to the main negative wire, which rested on top of the opposite side of the box from the positive main, was made by direct contact of the cross-bar at one end with the negative wire upon which it rested. The other end of the cross-bar rested upon a strip of wood which insulated it from the positive main.

The distance between the anodes and cathodes was therefore 1½ in. (4 cm.). Each box contained approximately a total of 3500 sq. ft. (320 sq. m.) each of anode and cathode surface. A detailed drawing of the whole arrangement is shown in Fig. 7.

The three boxes precipitated 150 tons of solution per 24 hr., resulting from the treatment of 50 tons per day of low-grade gold ore containing 0.20 oz. (5.8 g.) gold ($4.00) per ton. The solutions precipitated were kept at 70 deg. Fahr. (21 deg. C.). A low current density of 0.02 to 0.03 amp. per square foot (0.22 to 0.33 per sq. m.) of cathode surface was used. The voltage across the anodes and cathodes was 4 to 5.

Accurate figures are not available for the composition of the head solution precipitated, but the following are reasonably accurate estimates: Gold, 0.06 oz. (1.75 g.) ($1.20) per ton; cyanide, 0.01 to 0.05 per cent, and protective alkalinity, 0.03 per cent lime. The tail solution contained from a trace to a few cents in gold per ton.

When protective alkali was present there was no loss of cyanide in passing through the precipitation box, but when there was no protective alkali about 30 per cent of the cyanide was decomposed by the current. Presence of sufficient alkali also caused the iron dissolved at the anode to separate as ferric hydroxide, instead of the Prussian blue formed when there is insufficient alkali.

In this installation a special arrangement was made to run off the ferric hydroxide which collected on the surface of the solution, by shutting off the regular outflow and allowing the solution to rise above the tops of the high anodes and overflow momentarily from an auxiliary outflow carrying with it the ferric hydroxide. This operation required but two or three minutes upon each shift.

Anodes

Iron anodes were used with the Siemens-Halske process in South Africa, but lead, with a few exceptions, has been used in American practice. Various other anode materials have been tried but, everything considered, lead is the most satisfactory. One important advantage that lead has over many other anode materials is that it has a certain salvage value after it has served its useful life as an anode. It has been used both plain and peroxidized, but in general the peroxidized anodes have had the longest life.

Many investigators have reported certain anodes as being satisfactory when used in cyanide solutions, but these tests carried on in a small way have always covered entirely too short a period of time. The fact that an anode will stand up for two or three days of continuous operation in no way signifies that it will be satisfactory for a period of operation sufficiently long to make its use commercially economical.

Experiments with Acheson graphite have been most disappointing, notwithstanding the favorable reports made by other investigators who have tried out this material in a small way for short periods of time. The graphite anodes gradually become mushy and soft, and finally are useless. The reason for the failure of the graphite anode is not clear, but at one time I thought that it was due to the disintegration of the binding material used in the original carbon plates from which the graphite plates were made. In other words, the graphite resulting from the carbonaceous binding material of the original carbon plate does not appear to form a strong union with that resulting from the carbon proper of the plate.

The sheet-lead anodes are made up as shown in Fig. 8. The sheets of lead, after cutting to the desired size, are thoroughly cleaned and in one corner or on the two opposite corners of the short dimension of the sheet is cast a special connecting lug, ½ in. (1.3 cm.) in diameter by 4 in. (10 cm.) long, by means of a special iron mold which firmly clasps the plate at the point where the lug is to be attached. The lugs are cast from lead to which has been added a small amount of tin, to lower its melting point. The temperature of the lead used in casting the lug should be such that does not melt through the lead sheet, but still is sufficiently hot to form a thorough union with the lead sheet. A tongued wooden supporting strip is attached to the top of the plate by means of wooden pins.

The lead-rod anodes are made up as shown in Fig. 6. Lead rods of the form shown in the drawing are cast in special iron molds, observing the usual precautions for preventing the oxidation of lead during melting and the obtaining of a clean, smooth casting free from holes or spongy spots. Twelve of these rods supported in strips of pine drilled to receive them at the bottom, at the center and at the top, comprise an anode.

The bottom and center strips are 1.75 in. (4.4 cm.) by 1.25 in. (3.2 cm.), and are kept from shifting by means of wooden pins which engage the two outside rods. These also serve to prevent short-circuiting of the tin-plate cathodes, which have a strong tendency to buckle.

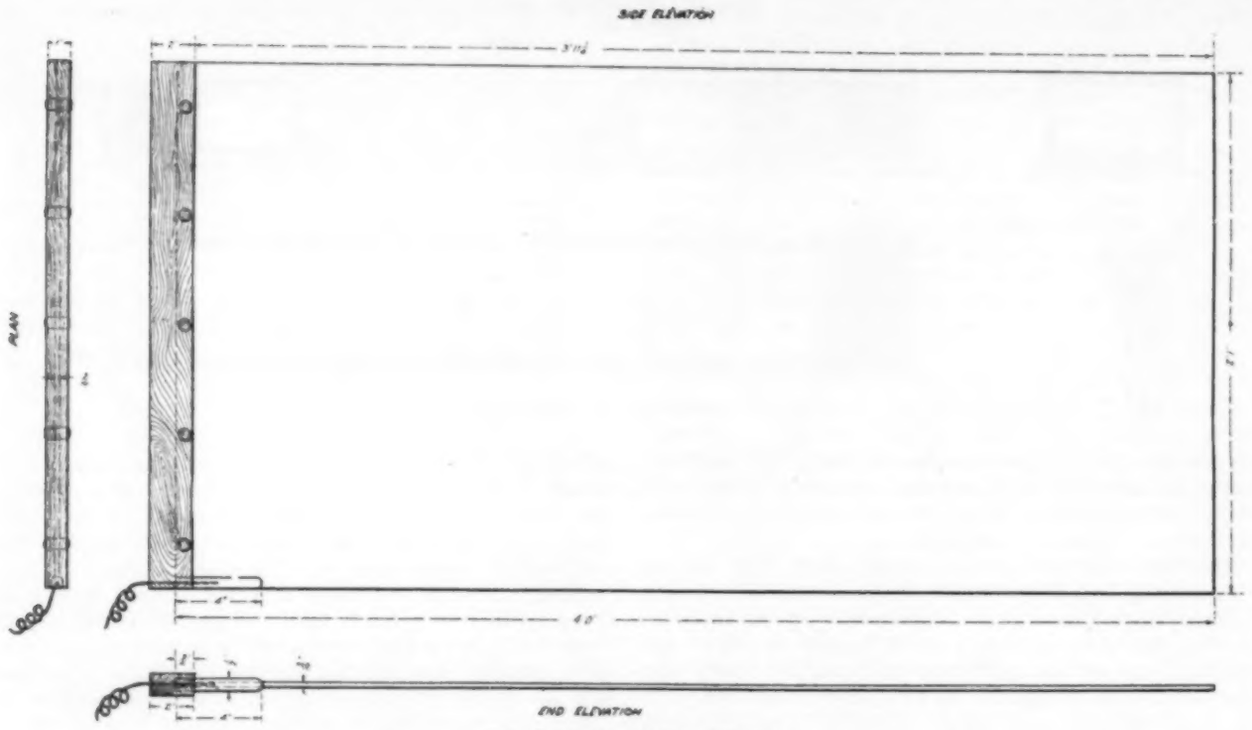

FIG. 8—LEAD PLATE ANODE

The top strip, 2¾ x 1⅞ in. (7 x 4.7 cm.), is counterbored to receive the large ends of the rods and supports the whole weight of the anode.

Electrical connection to the individual rods is made by soldering to them a No. 10 (2.5 mm. diameter) copper wire, which rests in a groove at the top of the strip. The continuation of this wire serves as the means of connecting the anode to the bus-bar.

The method of peroxidizing the anodes varied somewhat at the different plants. This operation was performed in a separate cell where a few anodes at a time were treated. The first step was to clean and brighten thoroughly the surface of the lead with a scratch brush.

At Minas Prietas a 1 per cent potassium permanganate solution was used and a current density of 1 amp. per square foot (11 per sq. m.). The time was 1 hr. or longer. During the operation the strength of the solution was frequently tested with a standard solution of ferrous sulphate, fresh permanganate being added to keep the solution as near 1 per cent as possible.

At San Sebastian various combinations were tried, but the practice that met with the most favor was to use an electrolyte containing 1 per cent of potassium permanganate, 2 per cent of sulphuric acid and a current density of 2½ amp. per square foot (27.5 per sq. m.). The time of treatment was 6 hr.

In passing, it is well to mention that it has been found in certain of the electrochemical industries that a denser and more satisfactory coating of the peroxide is formed if the operation is performed in an alkaline solution.

At Virginia City the peroxidized lead anodes had a life of about one year. At San Sebastian, on account of the higher current density the life was 8 to 12 months. At Minas Prietas the life was in some cases considerably over a year.

Cathodes

The lead-plate cathodes used at San Sebastian were constructed similar to the anodes, except that the sheet lead used was thinner.

An interesting feature of the lead cathodes used at the Reliance mill was that they were not ordinary rolled sheet lead, as had been previously used elsewhere for both anodes and cathodes, but were plates cast upon an inclined machine at the Trail smelter. The simplicity of the arrangement will be apparent from the drawing shown in Fig. 11. It was said that one man could make 500 cathodes in a shift.

The tin-plate cathode was made from medium-weight tin-plate, as shown in detail in Fig. 9. A piece of strap iron ¾ x ⅛ in. (1.9 x 0.3 cm.), whose length was the same as the width of the plate, was riveted to the top. The connecting wire was introduced between the plate and strap iron in such a manner that one of the rivets clamped it firmly in place.

Fig. 10 shows the tools used in removing the portion of the deposited metals which adhered to the cathodes. The double brush was used at intervals for removing the slightly adherent deposit. The manner of using was to set it over the cathode so that a brush was bearing on either side of the cathode, when the whole tool was pressed downward until the brushes had reached the bottom of the plate; this operation was repeated until the whole surface of the plate had been covered. When it was desired to make a complete clean-up, especially if the plates were old, it was necessary to remove them entirely from the box and scrape them with the triangular scraper, shown in Fig. 10.

By-Products of Electrolytic Precipitation

The by-products which appear to be inevitably produced during electrolytic precipitation and which are particularly troublesome when treating gold ores on account of the greater value of the metal involved, have always formed a most serious problem.

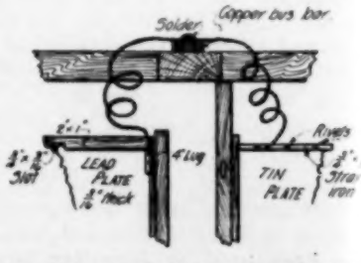

FIG. 8A—DETAIL OF CONNECTIONS (SEE FIG. 8)

In the Siemens-Halske process the Prussian blue, formed as a re-

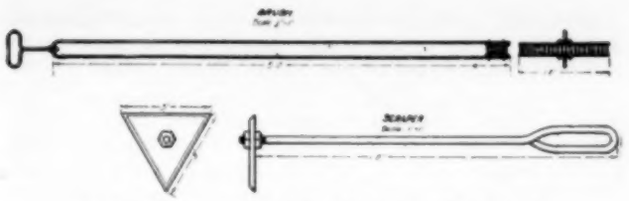

FIG. 10—TOOLS FOR CLEANING CATHODES

sult of the attack on the iron anodes, had to be subjected to special treatment for the recovery of the cyanide and gold which it contained. The disadvantages arising through the separation of calcium salts, which deposited upon the anodes and cathodes and in the bottom of the box, was early noted in the application of the process. It has been variously estimated that from 0.6 to 25 per cent of the gold had to be recovered from the by-products in the Siemens-Halske process.

At Minas Prietas, the precipitation of calcium salts was especially troublesome, and owing to the method of operating in which the non-coherent deposit of the metals was collected in the bottoms of the boxes, all of this eventually became intermingled with the precipitate. At this time the calcium salts, chiefly the carbonate, formed a large proportion of the precipitate.

TABLE V.—ANALYSIS OF LIME DEPOSIT

	Per Cent
Moisture	0.275
Loss upon ignition	6.060
Calcium oxide	41.580
Carbonic acid	31.120
Lead metallic	13.530
Iron oxide	0.540
Copper metallic	0.750
Sulphuric acid	4.400
Chlorine	0.670
Vanadium	0.095
Silver	0.126
Gold	0.003
	99.149

This made the precipitate more difficult and costly to smelt. In Table V is given a typical analysis of the lime deposit at the plant.

Lowered current density lessens the formation of the lime precipitate. In the course of time the lead anodes disintegrated to the extent that they had to be discarded. At first these were shipped to lead smelters, but later they were remelted locally, and after the separation of the dross, which had to be shipped to the smelter, were cast into slabs which were rolled by a special set of rolls into sheets which were used in making new anodes.

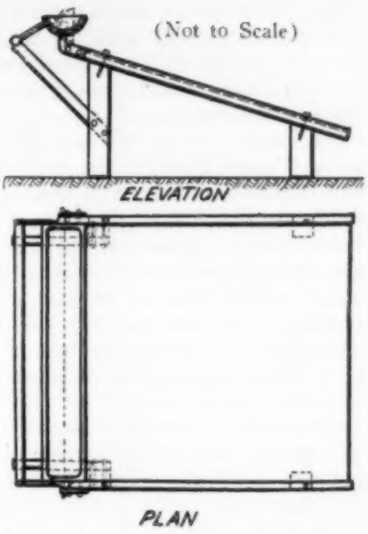

FIG. 11—CASTING MACHINE FOR LEAD PLATE CATHODES, TRAIL SMELTER, B. C.

At San Sebastian, where the gold and copper formed a coherent deposit upon the cathodes, it was possible to keep the deposit which accumulated at the bottom of the box separate from the deposited metals. However, the gold contained in this material was considerable, and at one time, when a considerable amount of it had been allowed to accumulate, it represented a great deal of value. The material more directly resulting from the disintegration of the anodes contained:

Lead	53.7 to 57.1 per cent
Copper	3.1 to 4.7 per cent
Gold	0.031 to 0.154 per cent

The fine material, or "lime" as it was termed, contained:

Silica	8 to 10 per cent
Alumina	22 to 28 per cent
Calcium oxide	15 to 21 per cent
Lead	Not determined
Copper	5.9 to 6.9 per cent
Gold	0.322 to 0.363 per cent

A small blast furnace and other refining equipment was installed for treating the by-products, but on ac-

FIG. 9—TIN PLATE CATHODE

count of the high cost of coke and other supplies and the lack of skilled labor, it was generally found more economical to ship this material to smelters in the United States. No attempt was made at this plant to reclaim the anodes.

At Virginia City the conditions were much the same as at Minas Prietas. A much lower grade precipitate was invariably obtained from the electrolytic boxes than from the zinc boxes on account of the calcium precipitate. At one time, as the result of careful sampling and assaying, it was estimated that the anodes contained about 10,000 ounces of silver. This came about by reason of the fact that the solutions contained chlorine. The source of this in the tailing was the salt which had been added during pan-amalgamation and, in the Tonopah ore, was due to the occurrence of part of the silver as chloride. The presence of the silver as chloride in the anodes was proved by solution in ammonia and the regular qualitative tests. In the earlier operation the disintegrated anodes were shipped to the smelter; but later it was found that by careful heating in suitable pots the greater part of the metallic lead could be liquated, leaving most of the gold and silver in the dross. The dross was shipped to the smelter and the lead cast into rods which were mounted in wooden supports and used as anodes.

Cost of Installation and Operation of Electrolytic Precipitation Plants

The cost of the six electrolytic precipitation boxes at Minas Prietas, Mex., was as stated in Table VI.

TABLE VI.—COST AT MINAS PRIETAS

Box construction	$10,000.00
Dynamos	3,000.00
Lead anodes	4,455.00
Tin plate cathodes	300.00
Wiring, switchboards and instruments	1,000.00
Labor	1,000.00
Incidentals	250.00
Total for six boxes	$20,005.00

At Virginia City, Nev., U. S. A., the cost of six standard boxes was as stated in Table VII.

TABLE VII.—COST AT VIRGINIA CITY

Box construction	$6,000.00
Dynamos	4,075.00
Lead anodes	4,080.00
Aluminum cathodes	*1,598.00
Wiring, switchboards and instruments	1,000.00
Labor	1,500.00
Incidentals	250.00
Total for six boxes	$18,503.00

*N. B.—Tin plate cathodes also used.

W. H. Wood, in 1899, compiled comparative figures of the cost of installation on the Rand, South Africa, of precipitation equipment for operating the Siemens-Halske electrolytic and the zinc-shaving lead-couple processes for a plant crushing 500 tons of ore per day. These are given in Table VIII.

TABLE VIII.—COMPARATIVE COSTS

	Zinc Lead Precipitation	Electrolytic Precipitation
Extractor house space	$792.18	$2,551.50
Concrete floor	296.46	928.26
Wooden containing boxes	558.90	3,159.00
Platforms	.....	703.00
Piping	109.35	257.58
Zinc (Cut.)?	299.62	.....
Dynamo	.....	1,083.78
Lead cathodes	.....	414.32
Frames for cathodes	.....	218.70
Iron anodes	.....	1,603.80
Anode bags	.....	148.23
Wiring and proportion of power	.....	1,044.90
Proportion of general charges	71.93	424.04
	$2128.44	$12,537.11

The only costs available for electrolytic precipitation are those given in Table IX, which are the average for six years' operation of the Minas Prietas plant.

TABLE IX.—ELECTROLYTIC PRECIPITATION COSTS AT MINAS PRIETAS

Power	$0.083
Labor	0.037
Supplies	0.031
Depreciation	0.04
Total per ton of ore	$0.191

It will be noted that both the cost of installation and operation of electrolytic precipitation is higher than zinc precipitation.

In the preparation of this paper I am indebted to the following gentlemen: Mr. Charles Butters for information upon electrolytic precipitation obtained through visits to his plants during the time that they were in operation, as well as through personal communications; E. M. Hamilton for information contained in his paper upon the practice at Minas Prietas, also personal communications; Charles P. Richmond for information contained in his paper upon the practice at San Sebastian, also personal communications; James Colbath for the details of the El Rayo practice; Richard Snell, H. Bulmer and Robert Wisnom for certain details of Virginia City practice; Douglas Lay for information contained in his paper upon the practice at the Reliance mill, and M. F. Perry for the detailed costs of installation and operation of the Minas Prietas plant. I have freely used the information contained in von Gernet's first paper upon the Siemens-Halske process, as well as that contained in other papers upon the subject which have been published at various times in the Proceedings of the Chemical and Metallurgical Society of South Africa. I also wish to acknowledge the faithful and painstaking work of my assistant, Mr. K. Pyle, in the preparation of the drawings.

Palo Alto, Cal.

Potash.—The American Potash Company of Long Beach, Cal., has been purchased by the American Produce Company, who will operate the plant for producing potash and cellulose from kelp on a large scale.

Stamp Mills.—The Allis-Chalmers Co. of Milwaukee, Wis., have issued Bulletin 1432-A describing their stamp mills and accessory machinery for free-milling gold ores. This includes specifications for complete ten-stamp and two-stamp mills.

Abrasive Grinding Wheels.—The Abrasive Material Co., Philadelphia, Pa., has sent us its catalog of abrasive grinding wheels, and also pamphlet describing Boro-Carbone, a new abrasive which it has recently placed on the market.

Precious Metals in Colorado.—The total value of the output of gold, silver, copper, lead and zinc in Colorado in 1914 was $33,460,126, according to the annual statement of the U. S. Geological Survey. Of the total, 59 per cent represented gold.

Resistance and Inductance of Wires.—A paper dealing with experimental determinations of the electrical properties of iron and bi-metallic wires used in telephone and transmission work has been issued by the Bureau of Standards as Scientific Paper No. 252. The author is Mr. John M. Miller.

The Johnson Engineering Co., Chicago, Ill., has just shipped to the Detroit Copper Mining Company, Morenci, Ariz., two of its largest diameter Marathon mills, which will do all of the regrinding in a plant of 1800 tons daily capacity. Five of the largest diameter mills were also shipped to the Burro Mountain Copper Company, Tyrone, N. M., for regrinding in the new plant of 3000 tons daily capacity.

Manufacture and Tests of Silica Brick for the Byproduct Coke Oven*

BY KENNETH SEAVER

It is rather gratifying that in the field of the manufacture of refractories, in which, as a whole, foreign practice has surpassed us, the United States stands pre-eminent in the making of silica brick. In some instances European manufacturers have refused point-blank to credit the possibility of making to such specifications as are common with us, until the actual completed shapes were shipped for inspection. But both the manufacturer and consumer in foreign lands are now frankly learning from us, and benefiting by our advanced practice in this field, as is evidenced by the orders placed here and the efforts put forth abroad to come up to the standard of American-made material.

In no metallurgical operation has the employment of silica brick spread with such leaps and bounds as in the building of plants for the carbonization of coal, particularly in the byproduct coke oven. To-day, in practically all types of oven, the structure above the floor level (except the facings) is of silica, while in the recuperative and in some regenerative types, its use is extended to that portion of the oven below the floor as well.

By reason of the fact that there are a number of classes of refractories of higher silica content than the usual so-called fire-clay brick, there is sometimes a misunderstanding as to the class of material referred to as "silica" brick. We here refer only to a brick having a silica content of 94 per cent or more and made usually from quartzite rock with a small percentage of lime as a binder. The term "silica" brick is sometimes rather loosely applied to brick from highly siliceous clays, or to a quartzite refractory of mingled fire-clay and quartz rock; but such products are not considered in this discussion.

Materials

The rock from which practically all high-grade silica brick are made is a true quartzite or metamorphosed sandstone, variously known as ganister, quartz, quartzite, sandstone, silica rock, etc.—the fields of major importance being in the order of their relative importance, those of Pa., Wis., Ala., and Col.

PENNSYLVANIA QUARTZITE

Pennsylvania probably manufacturers from 75 to 85 per cent of all the silica brick made in the United States, almost entirely from the quartzite found in the Medina and Oneida formation. While this formation extends across the State of Pennsylvania reaching into New York State on the north, and Maryland, Virginia and still farther toward the South, the quarries are practically all in Huntingdon and Blair counties, Pennsylvania. It is in this restricted area that the quartzite is at its best. Toward the north and toward the south it becomes softer, is no longer a true quartzite, or is too much impregnated with iron. The rock is also comparatively easily accessible in the counties mentioned, enormous quantities lying loose upon the surface. The particular occurrence of this loose rock has here been brought about by the Juniata River and its tributaries, as they have cut across the formation, wearing away the softer formations beneath, and leaving the Medina overhanging and exposed. In the course of time it has been weathered and broken up by the elements into the bodies of loose rock or "floes," as they are called, which may cover an area of a thousand acres or more. The depth usually varies from three or four to as much as 20 ft. Smaller floes formed in the same way may be found along the flanks of the mountains; but they nowhere compare in size with those formed where the measures have been cut transversely.

This rock is sometimes quarried out of the solid measures; but by far the largest amount used is that taken from the bodies of loose rock, as the latter is not only cheaper to work but is of more uniform quality. The white Medina formation is not composed entirely of quartzite, but is made up principally of sandstone with intermingled strata of quartzite. Thus where the formation has been broken up, the softer sandstones unsuitable for the manufacture of silica brick have tended to weather away and disintegrate, and have left the larger portion of the floes to consist of quartzite. When sandstone is mixed with the quartzite in the floes it is readily detected by its appearance, as it tends to become much more rounded and worn in weathering than does the quartzite, which maintains its sharp, angular edges.

The individual pieces in these floes range in size from those of fist-size to boulders weighing many tons. Probably 50 per cent of the rock occurs in pieces weighing more than 500 lb. The largest floes are found at the gorges of the Juniata River near the towns of Mount Union, Mapleton, Point View, Lewiston, etc.

It is interesting to note that the Oriskany sandstone, extensively used as glass sand, is found in close proximity to the Medina and that although exceedingly pure in its composition, its softness renders it unsuitable for high-grade silica brick. It consists of a mass of minute quartz grains loosely held together, and often so friable that it can be easily worn away by the hands. On the other hand, the Medina quartzite consists of quartz crystals cemented in a mixture of the same composition and of equal hardness with the crystals. The difference in the two types of rock when ground to sufficient fineness for the manufacture of brick is obvious. The harder stone gives splintery, angular fragments; the other, rounded grains. Experience shows that a hard rock like the Medina is required for a physically strong silica brick.

The white Medina which occurs in southern Pennsylvania near the town of Hyndman is there prominent and of a pure variety, but by no means as hard as the quartzite of Huntingdon County.

A representative analysis of the Pennsylvania quartzite is as follows:

	Per Cent
Silica (SiO_2)	97.80
Alumina (Al_2O_3)	0.90
Iron peroxide (Fe_2O_3)	0.85
Lime (CaO)	0.10
Magnesia (MgO)	0.15
Alkalies	0.40
	100.00

WISCONSIN QUARTZITE

This rock, much used in the manufacture of silica brick in the vicinity of Chicago, is known as the Baraboo quartzite from the town of that name where these measures appear, and near which are the quarries. The rock occurs in high cliffs throughout this region and is especially prominent around Devil's Lake. Here glacial action has excavated a deep, broad valley in the quartzite formation bounded on each side by cliffs several hundred feet in height. Their bases are covered with the loose rock split from the parent ledges. While the fragments are much larger as a whole than the weathered Medina of Pennsylvania, the quality by no means compares with the latter. Practically all the stone used in brick making is here quarried from the solid measures.

The marked peculiarity of this rock, as compared with the Pennsylvania quartzite, is its reddish color, which ranges from a light pink through a dark purplish red

*A paper read before the San Francisco meeting of the American Institute of Mining Engineers, September, 1915.

almost to black. It is often as red as the red Medina of Pennsylvania, a sandstone wholly unsuited for brick making. While the coloring of the Baraboo quartzite is due to iron, it is yet an exceedingly pure rock. The quartzite often becomes lighter in color as the quarry works in from the face of the cliff; but it is the exception to find white rock.

This Baraboo quartzite, estimated as some 4000 to 5000 ft. in thickness, is a metamorphosed sandstone, the sedimentary origin of which is indicated by its parallel bedding planes, by ripple marks, etc. Layers of quartzite are often separated by beds of quartz schist varying from a few inches to several feet in thickness. This schistose structure is due to the admixture during deposition of clayey material, which in the subsequent folding and bending of the rock has developed a slablike or schistose structure.

The United States Geological Survey explains the formation of the quartzite as due to the enlargement of the original quartz grains and to the deposition of interstitial quartz. It was the latter that carried the iron oxide, giving the characteristic color to the rock. Regardless of its color, the formation throughout shows the physical characteristics of hardness and strength, and the splintered and angular fracture in crushing, identical with the Medina quartzite of Pennsylvania. A typical analysis follows:

	Per Cent
Silica (SiO_2)	97.15
Alumina (Al_2O_3)	1.00
Iron peroxide (Fe_2O_3)	1.05
Lime (CaO)	0.10
Magnesia (MgO)	0.25
Alkalies	0.10
	99.65

ALABAMA QUARTZITE

The material used in silica brick manufactured in Birmingham, Ala., is found in the eastern part of that State. It is bluish white, and as hard and crystalline as the Wisconsin quartzite; average analyses show it to be equally pure. It occurs in lenses of varying size, in a sandstone formation. It is probably a quartzite formed by a leaching out of impurities from the parent rock followed by a crystal growth and additional deposit of silica from circulating waters. A typical analysis follows:

	Per Cent
Silica (SiO_2)	97.70
Alumina (Al_2O_3)	0.96
Iron oxide (Fe_2O_3)	0.80
Lime (CaO)	0.05
Magnesia (MgO)	0.30
Alkalies	0.31
	100.12

In Colorado and Montana both quartzite and sandstone are used to some extent in the manufacture of silica brick.

Method of Manufacture

QUARRYING

The methods of quarrying as necessitated by the mode of occurrence of the quartzite are so obvious as to need no description. The only point which is of importance is that of adequate supervision in the selection of the rock, whether it be from floes or from the solid measures. Whatever the location or occurrence of the rock, such care is essential; the easiest way to keep bad rock out of the brick is never to let it reach the plant.

GRINDING

After the rock broken to one-man size is received at the rear of the brick plant, it is put through some of the familiar types of crusher, which reduces it to practically the size of a 1- to 2-in. ring. It is then elevated to storage bins from which it may be fed direct

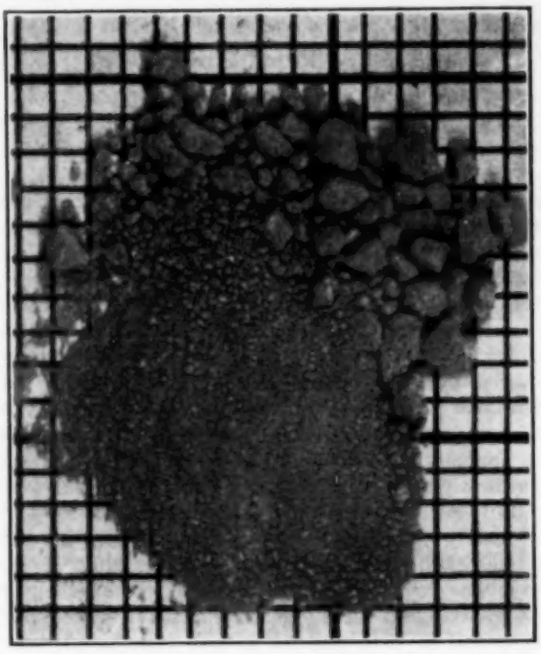

FIG. 1—MATERIAL FROM UNBURNED AVERAGE STANDARD 9-IN. BRICK

in definitely weighed quantities to the wet pans for grinding.

Into the wet pans, as the grinding proceeds, 2 per cent of lime, by weight, is introduced as milk of lime. When the brick is burned, the lime in combination with the other elements in the mass serves to give the necessary bond or physical strength. The quality of the lime has no small effect upon the resulting characteristics of the brick. The effect of an increase in lime from 2 to 3 per cent is not marked; but such as it is, renders the brick less able to withstand abrasion, as is indicated by rattler tests. The decrease in strength on further additions of lime up to 7 per cent is progressive, and becomes more and more evident with each addition. There is, on the other hand, a decrease in strength at 1.5 per cent of lime, and a marked weakness and friability as the amount of lime is still further diminished. As regards ultimate refractoriness, no marked effect is noticeable upon increasing the lime addition up to 3 per cent. From that point on the effect is clear in a lowering of the fusing point.

The fineness to which the charge in the wet pan is to be ground depends upon the nature of the brick to be made from it and is, of course, controlled by the time during which the charge is allowed to remain in the pan. In general, one charge, containing material for 200 brick, will be ground by approximately four pans per hour for standard 9-in. brick, while to obtain the fineness requisite for the intricate and difficult shape-work necessary in byproduct coke-oven construction, but two pans per hour may be possible. The rock is exceedingly hard, and a comparatively slight increase in the fineness of grind is accompanied by a seemingly disproportionate lowering of grinding capacity, and increase in wear on the pans.

The actual differences in grind, however, are much smaller than is often imagined. Figs. 1 and 2 are photomicrographs with a magnification of 2 diameters, the ground material showing on a background of cross-section paper with 0.1-in. spacing. The specimens are from pieces of green or unburned silica brick, crushed down after drying. Fig. 2 indicates typical fine-ground shape-work, while Fig. 1 is that of average standard

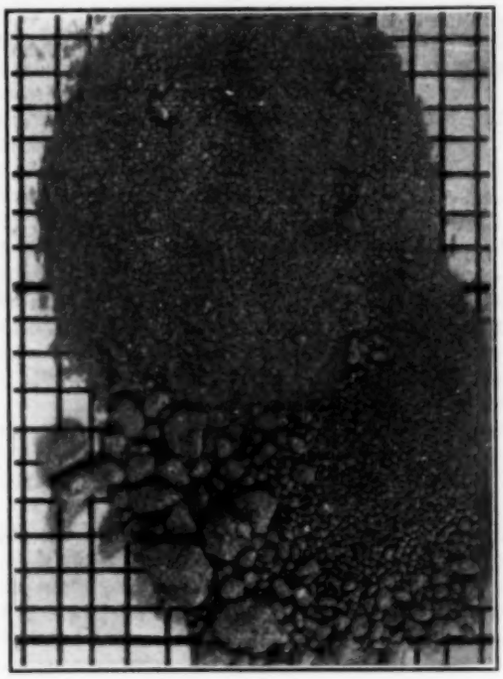

FIG. 2—SAMPLE SHOWING MATERIAL USED IN TYPICAL FINE-GROUND SHAPE WORK

9-in. brick. The difference between these is also shown in the screen analysis of the two.

It will be plainly apparent from the photomicrographs that the difference lies not so much in the actual size of either the coarse or the fine grains as in the relative proportion of fines. In the finer-ground brick the larger proportion of these is clearly evident. There can be no question that the general effect of increasing the fineness of grind is to improve the abrasive resistance of the brick, if it is properly burned. In general, the requirements as to grind contained in typical coke-oven specifications have worked no hardship upon the manufacturer; for the present-day standard of workmanship on byproduct oven shapes is such as to "carry its own grind with it," if the expression may be used. To secure the accurate filling out of the shapes, and the clean-cut accuracy everywhere essential, the grind is actually much finer than is requisite to satisfy the usual specifications as to that particular feature.

MOLDING

The ground silica material as ready to mold is of about the consistency of damp sand, but (possibly because of the extreme angularity of its grains) has less flow as it is manipulated. As a consequence, the molds must be filled by pounding with heavy plank beaters covered with metal, and all corners and shapes of irregular outline must be rammed in by hand, much as sand molds are made, but with much heavier blows. Unequal ramming, leaving too great a variation in the density of the green brick, will give unequal expansion in the kiln. It is by no means uncommon to lose hundreds of a new and difficult shape through such fire-cracking. To get proper molding and ramming requires the strictest supervision.

The wear on the molds due both to the attrition of the grains and the pounding is exceptionally severe. In view of the extreme accuracy required, steel molds are used exclusively for all high-grade coke-oven work. These molds are often decidedly complicated, and always expensive, as "loose sides" must be used throughout. Ground silica has, as molded, almost no physical strength, and can be taken from the mold only by allowing the sides to come with it, after which the sides are stripped.

After molding, the shapes are placed on pallets, on which the brick are dried either on the steam-heated hot floor or on rack cars in some form of tunnel dryer (Fig. 3). The brick are then ready to be set in the kilns. Although setting is a most important feature in the manufacture will not attempt to discuss it here.

FIG. 3—TYPICAL SECTION OF HOT FLOOR SHOWING COKE-OVEN "GUN" BRICK SHAPES IN PROCESS OF DRYING

BURNING

The vast majority of silica brick are burned with soft coal in the familiar type of round, down-draft kiln, varying in diameter from 30 ft. upward. Although practice may vary somewhat, and much depends on the size of the kiln, the following schedule of burning may be considered as fairly normal. Assume that the setting is complete and the doors are sealed. Fire is immediately started, and continued slowly and with slight acceleration for from nine to twelve days, during the last two or three of which the final temperature should be practically maintained. The kiln is then held at the required temperature, say a 26 cone (1650 deg. C., or 3002 deg. Fahr.) directly in front of the firebox, for another day, after which the fireboxes may be sealed and allowed to remain thus for approximately 24 hr. more. Cooling will require about as long as the burning proper; but this varies somewhat with atmospheric conditions.

It is obviously difficult to take temperatures in the interior portions of the kiln except by cones; but there are often most perplexing variations in this method. There are apparently certain conditions in the atmosphere of a burning silica kiln which alter the readings of standard cones to no small degree. Cones which give concordant results, week in and week out, in clay kilns are often decidedly erratic in registering the temperatures of silica kilns. It is by no means uncommon to witness cones of higher number go down, while next to them those lower in the series are either but slightly bent or apparently unaffected.

EFFECT OF BURNING ON COMPOSITION AND EXPANSION

In any discussion of the desirable qualities of silica brick there is frequently expressed the necessity that "all the expansion shall be burned out of them." This brings out the questions: to what is the expansion due; how may it be completely effected if that be possible; and by what means can that result be ascertained.

That the terms used in the discussion may be clear, let it be premised that all silica brick are of course made in molds smaller than the proposed size of the finished or burned brick, and that under the influence of heat a permanent expansion occurs, swelling the

brick from the mold or "green" brick size to the required or "burned" size. It is this expansion occurring during burning that it is desired to accomplish so completely as to preclude further expansion in subsequent use.

Now, in every burned silica brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. But it is argued that, if a silica brick is not properly burned, then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for.

It will be recalled that the silica brick under consideration is made merely by adding 2 per cent by weight of milk of lime to quartzite rock of the following composition, say:

	Per Cent
Silica (SiO_2)	97.80
Alumina (Al_2O_3)	0.90
Iron peroxide (Fe_2O_3)	0.85
Lime (CaO)	0.10
Magnesia (MgO)	0.15
Alkalies	0.40
	100.20

The representative silica brick produced may be assumed to have the following composition:

	Per Cent
Silica (SiO_2)	96.25
Alumina (Al_2O_3)	0.88
Iron peroxide (Fe_2O_3)	0.79
Lime (CaO)	1.80
Magnesia (MgO)	0.14
Alkalies	0.39
	100.25

What are the changes which occur as the siliceous mass is burned?

Certain of these are obvious, such for instance as the formation of a bond of one or more of the lime-silica compounds such as the meta-silicate CaO, SiO_2 or the ortho-silicate 2CaO, SiO_2. With the exception of the lime present, the peroxide of iron and the alumina, though small in quantity, are the preponderating impurities and there is undoubtedly formed a certain amount of silicate of iron as well as compounds of the lime-alumina series. Whereas the lowest melting point incident to any eutectic of the lime-silica system is 1436 deg. C. (2617 deg. Fahr.) and of the alumina-silica series is 1610 deg. C. (2930 deg. Fahr.), while that of the lime-alumina system is 1400 deg. C. (2552 deg. Fahr.), some of the eutectics of the ternary system CaO, Al_2O_3, SiO_2 melt at approximately 1170 deg. C. (2138 deg. Fahr.) and are thus among the first of the flux compounds resulting.

No less important, though less recognized, changes are occurring in the brick as it is heated. The evidence of the change, regardless of its reason, is easily indicated by the most casual measurement when brick are drawn from the kiln from time to time, as higher temperatures are reached. The final expansion will be very close to 38 in. per foot.

The valuable work done by Day, Shepherd and particularly by Fenner, on the Stability Relations of the Silica Minerals (*American Journal of Science*) give the results in a large field of laboratory experiment, which make possible a fair understanding of what occurs in the actual burning of the silica brick. Fenner's work was to establish among other things the relations between the minerals quartz, tridymite, and cristobalite, all silica minerals and any of which could be converted into any other by appropriate heat treatment. For the purposes of this discussion it is unnecessary to touch upon the alpha and beta phases of these minerals. In substance, the results under the conditions imposed by the experiments were that powdered quartz, if heated, remains quartz until a temperature of 870 deg. C. (1598 deg. Fahr) is reached, at which inversion point there begins a slow change to tridymite. If heat is still applied, there is no further change from tridymite until 1470 deg. C. (2678 deg. Fahr.) is reached, when inversion to the form of cristobalite has its beginning. There is no further change until the cristobalite may melt into a glass. Thus the stable phase of quartz (quartzite rock as applied to silica-brick manufacture) extends to 870 deg. C. (1598 deg. Fahr.). Between 870 deg. C. (1598 deg. Fahr.) and 1470 deg. C. (2678 deg. Fahr.) the stable phase is tridymite, while above 1470 deg. C. (2678 deg. Fahr.) cristobalite is stable.

As a whole it must be carefully noted that the changes from one mineral to another were exceedingly slow, particularly near the inversion points. So slow in fact were the alterations, that the melts were made in a flux of a large excess of sodium tungstate which acted as a flux or catalyzer. The powdered quartz or whatever might be the silica mineral under study was stirred into the molten bath of flux and the platinum container was then treated in a suitable furnace. This flux in no way entered into chemical combination with the silica but only aided and quickened its action, apparently furnishing a fluid medium through which the molecular rearrangement was facilitated. In spite of this, the slowness of the inversion, often requiring many hours, will bear emphasis in drawing conclusions regarding any parallelism between these experiments and the burning of brick.

A particularly noteworthy fact is that in the absence of a flux or catalyzer, the inversion from quartz to tridymite does not occur, but the inversion is direct to cristobalite, and there is a shift in the inversion point to as low as 1000 deg. C. (1832 deg. Fahr.) while at 1300 deg. C. (2372 deg. Fahr.) the change is very rapid.

This entire absence of tridymite in quartz heated well above 870 deg. C. (1598 deg. Fahr.) the quartz-tridymite inversion point, although difficult of explanation, is apparently proved. As indicative of the time element requisite for such an inversion, in one experiment fine-ground quartz heated 108 hr. at 1250 deg. C. (2282 deg. Fahr.) without flux gave but a small percentage of inversion while with 90 hr. at 1360 deg. C. (2570 deg. Fahr.) approximately two-thirds became cristobalite.

In considering these changes in the crystalline form of silica, some of the characteristic properties of the three will not only make clearer the method of distinguishing them, but also throw some light on the reasons for the effects these inversions exhibit.

Quartz has a specific gravity of 2.65, tridymite of 2.27, and cristobalite of 2.33. Each has its characteristic crystal form, the quartz being usually in the familiar pyramids, prisms or combinations of these. Tridymite often occurs in thin tablets or plates hexagonal in shape. Cristobalite is exceedingly irregular in the development of its crystal form but often develops octahedral caps along the closely interlacing axes of elongation.

It is the difference between the indices of refraction of these minerals, however, that furnishes the greatest aid in distinguishing the one from the other.

Tests.—As a starting point from which to work in determining the effect of burning, a set of sample silica brick were taken from one kiln, all from the same zone

and from a kiln of absolutely normal burn. In this instance they were first burned at a point about 8 ft. from the wall and 6 ft. from the bottom. This sample was divided into three parts, one to be kept unaltered, one to be reburned under the same conditions, and a third to have two reburns. In addition to this, pieces of quartz rock received one and two burns respectively, with the view of obtaining the same data.

The primary object was to determine, if possible, how far the inversion from quartz to cristobalite had progressed under the successive conditions. Each burn was made as nearly uniform as practicable, and in each succeeding burn the effort was made to maintain identical conditions. At that part of the kiln where these brick were burned the temperature reached 1540 deg. C. (2804 deg. Fahr.), and they were subjected to that heat for approximately 40 hr.*

In general, the chief optical constant serving to distinguish cristobalite on the one hand from quartzite and calcium silicate on the other was the index of refraction. The slow ray of quartz has a refractive index of 1.544 while the index of the fast ray of cristobalite is about 1.485. The immersion method was used to determine the indices of the grains, the powdered brick being immersed in a liquid of index 1.51. By this method it was possible to distinguish between the higher-refracting quartz and calcium silicate and the lower-refracting cristobalite.

With the aid of a *camera lucida*, the fields were sketched on very thin tracing paper. The grains, as represented on the paper, were then cut out and weighed for each constitutent. Aside from the fact that the composition of different fields may vary, other errors are entailed in the sketching, cutting out and weighing so that a variation of 5 per cent is justifiable. In these determinations the fields of the same specimen checked to within 5 per cent maximum and 1 per cent minimum error.

Results.—The tabulated data are as follows:

Specimen	Field	Quartz and Calcium Silicate, Per Cent	Average, Per Cent	Cristobalite, Per Cent	Average, Per Cent
Lot 1	1	24.87	.....	75.13	.....
(1st burn)	2	20.42	22.65	79.58	77.35
Lot 2	1	14.31	.....	85.69	.....
(2d burn)	2	19.96	17.13	80.04	82.87
Lot 3	1	17.44	.....	82.56	.....
(3d burn)	2	14.60	16.02	85.40	83.98
Quartz rock	1	50.08	.....	49.92	.....
(1st burn)	2	51.05	50.57	48.95	48.95
Quartz rock	1	32.70	.....	67.30	.....
(2d burn)	2	30.06	31.38	69.94	68.62

A number of interesting deductions may be made even from these preliminary tests. The maximum change takes place at the first burning, followed by a further appreciable inversion at the second, and practically no change at the third.

The grain structure of the reburned silica brick may throw some light on this subject. Here the grains may be seen to be essentially cristobalite, with the quartz occurring as an inclusion. The last stages of the inversion are apparently exceedingly slow within the protecting envelope.

The smaller percentage of inversion in the case of the quartz rock as compared with the brick is very noticeable, and is undoubtedly due to the grinding of the quartz in the manufacture of the brick. This is wholly in line with Fenner's laboratory experiments, in which he found the influence of comparative grinds to be slight if the silica mineral were heated with a flux, but to be marked if the material were heated without a flux. Fine grinding greatly hastened the inversion. The greater freedom of movement in expansion as compared with the exceedingly dense quartz body is probably the controlling factor, rather than any difference in the penetration of the heat.

The part played in the inversion (if any part is played) by the lime and its compounds present, is of much interest. It appears logical that the greater fluidity conferred by the fluxes present should have an influence; but it must be borne in mind that, as compared with previous experimental work on silica minerals, the amount of flux present in the brick is exceedingly small, and that definite chemical compounds are formed in combination with the brick body. In Fenner's work the sodic tungstate formed no such chemical compounds, and remained unchanged at the conclusion.

The fact moreover that no tridymite was determinable in either the burned brick or the calcined quartz indicates that the burning of silica brick is closely comparable to heating the quartz without a true flux or catalyzer; for did the impurities of the rock and the added lime act in that manner, tridymite would undoubtedly be distinguishable.

Thus far, we have assumed the changes in crystal form to be desirable, because of their effect on the expansion of the material.

EFFECT OF BURNING ON PHYSICAL STRENGTH

It is also of interest to note the effect of the reburning on the physical strength of the brick. To determine this, the following tests were run:

CROSS-BREAKING OR MODULUS OF RUPTURE TEST

Number of Burns	Mark	Breadth of Brick, Inches	Depth of Brick, Inches	Weight Causing Fracture of Brick, Pounds	Length Between Supports, Inches	Rupture, Pounds	Average, Pounds
1	9 A	2.46	4.40	3,550	6	671	.....
1	9 A	2.42	4.40	3,000	6	567	624
2	9 B	2.50	4.50	4,140	6	730	.....
2	9 B	2.50	4.50	5,000	6	889	809
3	9 C	2.06	4.50	4,500	6	971	.....
3	9 C	1.96	4.50	4,550	6	1,031	1,001

CRUSHING TEST; WHOLE BRICK CRUSHED FLAT

Number of Burns	Mark	Dimensions, Inches	Area, Square Inches	Load, Inches	Crushing Strength Per Sq. Inch, Pounds	Average, Pounds
1	9 A 2	9.00 by 4.40 by 1.96	39.60	176,800	4,465	.....
1	9 A 2	9.04 by 4.44 by 2.04	40.14	167,000	4,161	4,313
2	9 B 2	9.10 by 4.46 by 2.02	40.58	186,500	4,596	.....
2	9 B 2	9.10 by 4.40 by 2.06	40.04	168,000	4,196	4,396
3	9 C 2	9.06 by 4.42 by 2.00	40.04	195,000	4,870	.....
3	9 C 2	9.12 by 4.46 by 2.06	40.68	168,000	4,130	4,500

The increased strength of the brick is evident.

Conclusions

In considering whether or not all the expansion possible is burned out of the brick, it is necessary to determine first the limits and then how closely they are approached.

Day, Sosman and Hostetter say that the true specific volume of cristobalite as compared with quartz at 20 deg. C. is 13.4 per cent greater. Assuming a green brick to be wholly pure quartz (100 per cent SiO_2), and the brick burned from it wholly cristobalite, the volume increase would then be 13.4 per cent. This theoretically possible expansion is diminished by the effect of the impurities present in the actual brick, and by the resulting compounds formed. For instance, 4 per cent of impurities would produce a much larger proportion of compounds to which this expansion formula does not apply.

An expansion of, say 3/8 to 7/16 in. per foot, to which byproduct coke-oven silica is burned, thus provides for somewhat more than 13 per cent increase in volume.

It must be borne in mind, however, that by quick burning the brick may be easily expanded by double or

*The heartiest appreciation is here expressed to A. A. Klein, Petrographer in the U. S. Bureau of Standards at Pittsburgh, for his courtesy in making these determinations that they might be concluded in time for presentation here. The quality of Mr. Klein's work needs no comment.

treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick by a too sudden rise in temperature during burning. In the early stages of firing the heat must be raised very slowly. In general the conclusion is justified that the present good commercial practice carries the results of the quartz-cristobalite inversion to the greatest degree economically possible and that the measure of the expansion of the brick affords a closely approximate measure of the inversion—providing only that the brick are sound.

The short series of tests outlined above are given with full realization that, while indicative, they are by no means conclusive. They were but the preliminary runs, necessarily made somewhat in haste, but giving data which permit intelligent plans to be made for the more comprehensive tests now under way. There is much to be learned as to the effect of both time and temperature as well as the influence of pressure, which may produce a shifting of some of the inversion points. Judging from the work thus far done, it is impossible to expect the full inversion of quartz to cristobalite under any practical operating conditions of manufacture.

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Unstable States in Arc and Glow*

BY W. G. CADY

In solid and liquid conductors, in which Ohm's law is applicable, unstable states, where a small variation in voltage produces a large change in current, are almost unknown. Exceptions occur only when a critical state in the conductor is reached, as, for example, in an iron wire at a certain temperature, or in an aluminium valve cell at break-down voltage.

Characteristics of the gas discharge.—When an electric current passes through a gas, the number of available carriers of the discharge increases with increasing current at a very rapid rate. In a popular sense, an ionized gas is a most "unstable" conductor, having a resistance by no means amenable to Ohm's law. In the language of dynamics, however, a gas discharge is normally stable in the sense that there is a restoring force to damp out small disturbances. The nature of this force, as well as the criterion for stability of any discharge, can be understood best by reference to the characteristic curve of the discharge, Fig. 1.

This curve shows qualitatively the relation between current and voltage in the discharge between two fixed electrodes. The nature and pressure of the surrounding gas are supposed constant, except as modified by conditions at the electrodes themselves. All of the common types of discharge are represented on this diagram. With a sufficiently high emf. it is theoretically possible, by decreasing the external resistance, to pass through all the stages in succession. Practically certain portions are unattainable, and different sources of supply are necessary for different forms of discharge. In any case, however, the curve is characteristic of the discharge apparatus and not of the supply.

The first part of the curve, *OA*, represents the saturation current, which soon assumes a value that is practically constant over a great range of voltage. This current is carried by the few free ions that are always present. As the voltage increases, the charged particles are sufficiently accelerated to form new ions by impact. At this point the current begins to increase, and a faint glow appears. The point *A* represents the sparking potential. In the present case the external resistance is supposed to be too high to admit of sparking.

AB is the glow discharge proper, in which the rate of ionization is so great that the voltage across the gap begins to drop. If the cathode is small, the curve may rise again before *B* is reached, owing to the abnormally high cathode drop.

Since the drop at the cathode is much larger than that at the anode, the cathode becomes relatively hot. As *B* is approached, the expenditure of energy at the cathode becomes so high that the cathode rapidly becomes incandescent. Here a new phenomenon appears, namely, the emission of electrons from the hot cathode, according to Richardson's law. These greatly increase the

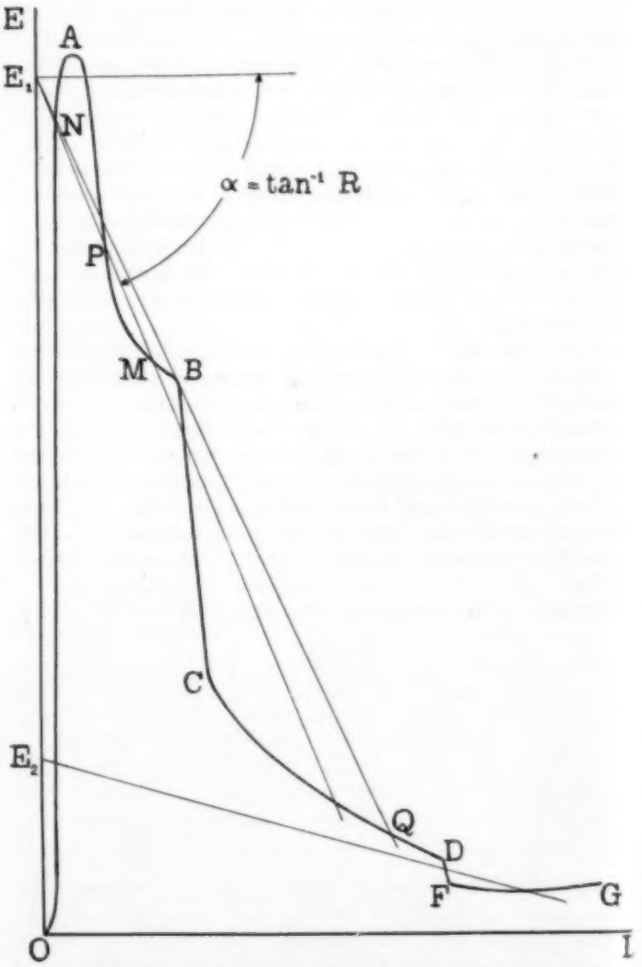


FIG. 1—CHARACTERISTIC CURVES OF GAS DISCHARGE. *OA*, SATURATION CURRENT; *AB*, GLOW; *CD*, ARC, FIRST STAGE; *FG*, ARC, SECOND STAGE

ionization close to the cathode, and the positive ions thus produced heat the cathode by impact still more. If the cathode metal is volatile enough, it begins to vaporize, thus lowering the cathode drop as well as the resistance of the gas.

The discharge quickly resolves itself into an arc, in which the negative base is concentrated to an intensely bright spot, the current is carried chiefly by the vapor from the cathode, but in which the mechanism of the discharge is essentially as before. The portion *BC* of the curve cannot be observed. Over *CD* the discharge is an arc as far as the cathode is concerned, but since less energy is being expended at the anode, at the latter we may still have a glow discharge. The positive base of the discharge is still diffuse and faintly luminous. This is the *first stage* of the arc.

*A paper read before the New York Section of the American Electrochemical Society on November 11 in joint session with the Illuminating Engineering Society.

With increasing current the anode is brought to the boiling point. This causes a decrease in the anode drop and a further increase in conductivity of the gas, so that the portion *DF* is quickly passed over. *FG* is then a full arc, or *second stage* arc. Exceptions to this typical sequence of events will be described below.

Quantitatively it is impossible to make the diagram give a correct idea of the voltage and current relations, since the current at *G* is in reality many thousands of times greater than at *A*.

Stability of the discharge.—While theoretically it is possible to maintain a steady discharge at any point of the entire curve, practically the transition from one portion to the next is always discontinuous. As the voltage or the external resistance is varied, a critical point is always reached where the discharge is absolutely unstable, changing abruptly to a stable region on another part of the curve. Still, with any given supply emf. and external resistance, it is always possible to predict between which two points the discharge must change. The method of procedure is largely due to Kaufmann.[1]

Let us suppose that the supply emf. is measured by OE_1 in the figure. If *M* is the point on the curve at which the discharge takes place, then since the ordinate of *M* is the voltage consumed by the discharge, it is evident that the external resistance *R* is equal to tan α. If *R* is decreased, α must decrease and *M* will move to the right along the curve.

Kaufmann showed analytically that a stable discharge is only possible when the line through E_1, or the "resistance line" as it is called, *slopes more steeply than the charactertistic curve* at the point in question. Thus the discharge is stable at *M*, and, starting with zero current, it would also be stable at *N* with the same external resistance, but not at *P*. A glow at *P* could only be obtained with an enormously high emf. At *B* the discharge is unstable; this is the first of the unstable states to which attention is directed. As soon as the point of tangency is reached, the discharge changes abruptly to a first-stage arc at *Q*. If now the external resistance is increased, the discharge does not change back to a glow along *QB*, but owing to the vaporization of the cathode it remains an arc until the point of tangency at C is reached. Then if, as is usually the case, the discharge cannot change to a stable glow, it goes out.

Similar points of instability exist at *D* and *F*. If the supply emf. is low, as at E_2, not even the first stage of the arc is possible, but only the second. This is the case with most arcs, operating with low emf. and small external resistance, and lighted by touching the electrodes together.

The physical significance of Kaufmann's criterion may be seen from the following considerations: If the resistance line slopes more steeply than the characteristic curve, then if through any chance the discharge at a given instant happens to take place at an abnormally high current, it cannot persist at this current, since the voltage available for the discharge is limited by the external resistance, and the steep slope of the resistance line brings the operating point too low for the discharge to be maintained; hence the current settles back to its normal value. Similar reasoning will show that if the resistance line slopes less steeply than the characteristic curve, then any slight disturbance will grow until the discharge goes out or becomes stable at another part of the curve.

In a general way this sort of stability is, of course, analogous to the mechanical stability of a pyramid, which is stable or unstable according to whether it stands on its base or is balanced on its point. A more exact analogy would be furnished by a hydraulic model, in which the discharge tube was represented by a valve so contrived as to open wider the greater the flow of water through it. This would provide the "falling characteristic." If then the water were supplied from a vessel of variable height, and regulated by a valve at the supply, it is evident that at high pressure and with supply valve nearly closed any given flow of water would be maintained more stably than if the pressure were low and the supply valve wide open, for in the latter case a much greater percentage of the total "resistance" would be in the valve of "falling characteristic."

Kaufmann applied his criterion of stability with great success not only to glow and arc, but to the point discharge, to a discharge gap ionized by an external source and, most remarkable of all, to the case of a discharge having a capacity in parallel with it. This came pretty near being the discovery of the singing arc, which, by a coincidence, was discovered experimentally by Duddell in the same year.

The carbon arc.—I have obtained a glow discharge and both stages of the arc between carbon terminals, using an emf. of 460 volts, in nitrogen at reduced pressure. Air was then admitted and the current increased until the hissing state was also reached, making four forms of discharge with the same electrodes. On the characteristic curve for carbon electrodes the hissing point would be represented by a kink similar to that at *DF* in Fig. 1, but situated between *F* and *G*. The laws governing the current-voltage relations on transition between normal and hissing carbon arcs are the same as those discussed above.

First and second stages of arc. Effect of impurities.—The unstable state occurring between the first and second stages of the arc has been investigated by Arnold and the writer. It is on the first stage that the ordinary mercury arc burns, having an anode of iron or graphite. The change from first to second stage has now been observed with anodes of a large number of metals, though in some cases reduced gas pressure or exclusion of oxygen was necessary.

An instability in the discharge is sometimes occasioned by traces of impurities on the surfaces of the electrodes. It is well known that the glow tends to change to an arc more readily if the metallic cathode is contaminated with traces of oxide. This is related, of course, to the emission of electrons from hot oxides. At the anode in a glow or a first stage arc, traces of impurities serve as nuclei around which the discharge tends to become concentrated, though it is still a glow. This effect may become so pronounced as to be taken for a new type of dischage,[2] especially since the voltage suffers a sudden drop when the concentration sets in.

An unusual type of arc.—A very unusual type of discharge is that in which, as the current increases, an arc appears at the anode while there is still a glow at the cathode. Those who define the arc as essentially a cathode phenomenon will not be disposed to admit that this is an arc at all, yet it has every characteristic of an arc at the anode, and the appearance here does not alter further when the arc phase sets in at the cathode. The writer has encountered this form of discharge with anodes of C, Ag, Cu and Hg, the cathode in most cases being of silver. The condition is decidedly unstable.

Is an arc possible from a pure metal?—The writer observed some years ago that a cathode of pure silver may become heated by the glow discharge in nitrogen nearly to the boiling point without the formation of an arc. At currents as high as 1.3 amp. the

[1] Kaufmann, Annalen d. Phys. 2, p. 158, 1900.

[2] Hagenbach, Phys. Zeitschr. 12, p. 1015, 1911.

discharge is still a glow, though the molten metal is white hot and vaporizes so intensely that clouds of silver "steam" fill the discharge tube. This is important evidence that incandescence and vaporization of the cathode are not enough of themselves to cause the formation of an arc. Whether even at the boiling point of silver the glow would still persist if no impurities were present cannot be stated. In the present tests, at the highest currents the cathode mantle always grew so large as to encroach upon the fringe of impurities that inevitably surrounded the molten mass, and then an arc immediately formed. The admission of a small quantity of oxygen also precipitated an arc.

Intermittent discharge. Corona.—Both glow and metallic arc, as well as the discharge from points, are sometimes found to be intermittent. For this to be the case, the current must be relatively small and the capacity of the external circuit considerable. There is usually a certain critical external resistance at which the intermittence vanishes, as shown by the cessation of sound in a telephone connected to the circuit. The phenomenon is closely related to the use of the arc as a generator of electric oscillations. The Cooper-Hewitt mercury interrupter is a notable example of intermittent arc.

In a strong magnetic field an arc can also be made intermittent. In this case the instability is due to the lengthening of the column of vapor by the magnet, causing the voltage to rise above the point where the discharge can be maintained. Owing to the incandescence of the cathode, the discharge quickly relights and the operation repeats itself.

The corona discharge from high voltage transmission lines is similar to a glow discharge, and is said to be often intermittent. It exhibits the characteristic cathode and anode phenomena, as well as the tendency to concentrate at certain points which was mentioned above. Since the air serves as one electrode, the corona has in some respects more in common with the discharge from points than with the typical glow.

Rotations at the anode.—Two effects will now be described, illustrating the instability of gas discharges as certain currents. The first of these has to do with the anode, the second with the cathode.

In the iron arc in free air the discharge normally takes place between globules of molten oxide. The second stage sets in at a little over an ampere. A close examination shows that the bright spot at the positive end of the arc is in rapid rotation, describing a minute ring. As the current increases this form of rotation suddenly changes to one in which the ring is still smaller, then at about 1.8 amp. a large ring takes its place, which presently yields the stage to a smaller ring again.

Each of the four types of rotation has its characteristic diameter and speed, and produces a faint note of a certain pitch. Oscillograph records show that the current and voltage undergo corresponding changes. Reasons have been given elsewhere[3] for believing that these movements are due to a reduction of the oxide at the positive base of the discharge, causing the base to wander to regions richer in oxygen. The changes from one type to the next are probably due to modifications in the composition of the oxide, brought about by increasing temperature. I have found similar effects to a slight extent in nickel and cobalt. These results have a bearing on the use of the iron arc as a wave-length standard in spectroscopy.

Glow-arc oscillations.—The effect at the cathode to which reference has been made is of the nature of a rapid pulsation back and forth between glow and arc. To obtain it, the supply emf. must be high, and the gas should be hydrogen at atmospheric pressure, mixed with a little hydrocarbon vapor. The electrodes may be of any metal, placed very close together, the discharge current being of the order of 1/10 amp.

When the discharge is a glow, a spot on the cathode becomes heated to such a degree that a small arc forms, but the voltage then drops so much and the current increases so little, owing to the shape of the characteristic curve, that the energy is too slight to maintain the arc, and immediately the glow sets in again. Such pulsations between arc and glow are of a frequency anywhere between 1 and 250,000, depending on various conditions. I have succeeded in converting 66 per cent of the power applied to the discharge into high frequency oscillations of this type. Unfortunately, the nature of the discharge is such that the conversion of large amounts of power by this means is impossible.

In these experiments no capacity was connected in parallel to the discharge, though the adjacent parts of the circuit acted as capacities for the high frequency currents.

The singing arc.—In the singing arc, including the Poulsen arc used in radio-telegraphy, a capacity and self-inductance in series are placed in parallel with the arc. The effect of the capacity is to drain energy from the arc when the current in the condenser circuit is in one direction, and to strengthen the arc current when in the other direction. Continuous oscillations are thus maintained, the condenser circuit receiving energy and timing its own impulses in a manner analogous to the balance wheel or pendulum of a clock.

The point of chief interest here is that, as Kaufmann, Simon and other have shown, such oscillations are possible only when the arc (or glow) possesses a falling characteristic. Simon has found that the alternating-current, or "dynamic," characteristic differs from the direct-current, or "static," characteristic, the difference being due to a sort of hysteresis, and that the result of this is that at high frequency the dynamic characteristic tends to change from a falling to a rising type. This limits the attainable frequency, but by reducing the hysteresis and choosing such conditions as will make the characteristic fall as steeply as possible, frequencies of over a million are easily reached.

If the oscillations are weak, the direct current through the arc simply fluctuates. This can occur with the carbon arc. If the oscillations are stronger, they put the arc out at each cycle, and before the arc relights the condenser becomes charged. The oscillating arc used in radio-telegraphy is of this type. A third type, in which the oscillations are powerful enough to make the arc relight in the opposite direction, produce a coarse tone and are not much used.

Effect of second harmonic.—When the oscillations are of the second type, extinguishing the arc at each cycle, the wave form is very complex. The second harmonic is especially prominent. As an example of oscillations of this type of audible frequency, the following experiment may be described.

A carbon arc is made to sing by connecting a large capacity and a self-inductance coil of low resistance in parallel with it. A second coil is connected to a capacity such that the natural period of the secondary circuit is nearly the same as that of the circuit connected to the arc. When the two coils are brought near together, the pitch of the arc changes, owing to the effect of close coupling. If the pitch falls, it can be made to reach an unstable value from which it suddenly jumps up a whole octave, the discharge remaining stable at this higher pitch.

The explanation of this is, that when any two oscillating circuits are closely coupled there are two possible frequencies, one higher and one lower, in either of which

[3] Phys. Rev. 2, p. 249, 1913.

oscillations may take place. At a certain stage of coupling the higher frequency is just an octave above the lower. If now the arc is on the lower frequency and has a very strong second harmonic, then if resonance is also possible at the octave above, the harmonic may be reinforced to such an extent as to assume the rôle of fundamental frequency itself. The pitch then rises to the higher octave.

It is hoped that the phenomena that have been described may serve to illustrate the unstable nature of some forms of gas discharge, as well as the very diverse ways in which the instability asserts itself.

With respect to practical conclusions, the following may be said: for stability, an arc should be operated with supply emf., external resistance, and current all large. Or if the discharge is to be a glow, a region on the characteristic should be chosen at a safe distance from a critical point, having due regard to the supply emf. For oscillations, a steeply falling characteristic, small current, large emf. and artificial cooling of the electrodes are important factors.

Wesleyan University,
Middletown, Conn.

Notes on Chemistry and Metallurgy in Great Britain

(From our London Correspondent)

The General Situation in Great Britain

When a patient is in the throes of fever, when all normal functions are interrupted, and when several organs overstrained perform new functions, comparative standards by which to mark progress are not easily chosen. The world has had biggish wars before this, but none so big or so complex in economic and industrial aspects. Reckoned against all previous prophesies it was never considered possible that European civilization could have fought for so long or maintained such huge armies without utter collapse. Yet the collapse is far off when one or other group will throw up the sponge.

You, on your side should be seeing most of the game, but as you only see it through the filtering medium of censorships it is possible that the little we do see here of our own condition is of more utility to us as a combatant than the wider survey of a neutral.

Despite casualty lists, despite increasing indebtedness, despite the increasing conversions of factories and foundries to make munitions for cur armies, the general outlook is one of confidence. Of course, there is the masked anxiety of those whose relatives are in the trenches, but it is well masked.

Blunders, of course, there are, but administratively we are much more likely to suffer because those in authority are honest fools than because they are clever rogues. With free ports and open seas raw materials and foodstuffs cause us no anxiety. The harvest season being almost over, recruits from the industrial class which can best spare them, will come forward. Despite the ill-advised advertisement given by the press to certain cranks the labor situation has greatly improved.

But in one respect there is cause for alarm, an alarm as to the consequences of the present high wages, good employment, and resultant present extravagance. To a certain extent the latest budget will check the tendency to extravagance, and for the rest, stern necessity will in due course intervene, and forcibly deal with the fruits of an unhealthy and abnormal activity.

The British Association Meeting

Among the many papers and sectional presidential addresses that by Dr. H. S. HELE-SHAW to the Engineering Section stands out with much prominence. Dr. Hele-Shaw is among those, who, while giving full attention to present needs is also giving careful consideration to the problems which will follow the end of the war. He says, "I do not believe the Germans despise us for our want *per se* of the application of science to industry. I do not think they have much reason to; but what they do despise us for is the want of co-ordination, which, I venture to say, amounts to positive slackness, which they are keen enough to observe permeating the whole of this country. They see different sections, instead of being united for a common end, quarreling with each other, filled with mutual suspicion and distrust, with apparently no common bond of union, and whereas the German is proud of the Fatherland, he sees in this country large numbers who seem, either through self-consciousness or ignorance, to be ashamed to mention the subject of the British Empire, or, what is worse, to acknowledge that any love of their country is or could be a mainspring and incentive to strenuous effort. . . .

"When peace is concluded, it will only be a prelude to another war, and a war which will recommence with a far greater energy on the part of our enemy than before, viz., the war of commerce—and the latter will be almost as serious for us as the more sanguinary one. This will be so even if we are victors, for 'after a long war the difficulties of the victors are often greater than those of the conquered. The conquered have their attention directed to the reparation of losses and are inspired by a patriotic desire to submit to losses for the sake of their country. The victors are in the frame of mind which expects everything to be easy.'

"Remembering how swiftly we relapsed into national ease at the end of the Boer War, it behooves every man who can do so to take his share in making ready for the terrific struggle Germany is certain to put up in the arts and manufactures. We shall better appreciate what this competition will mean if we consider the marvelous stride made by Germany during the last half-century in the arts and manufactures. It is undoubtedly in the matter of scientific organization even more than the organization of science that Germany has achieved such wonderful results, and it is, therefore, in this direction that we must leave no stone unturned if we wish to have any chance of holding our own in the future."

Also of importance was Prof. W. R. SCOTT'S presidential address to the Economic Science Section—speaking of the probable situation at the end of the war, Professor Scott pointed out that there may be a short trade boom (arising out of an attempt to restore some of the material ravages of war), but the joint demand from it and from the trades reopened is likely to be considerably less than the huge present expenditure of manufacturers for war. Thus the unemployment occasioned by dislocation of industry through hostilities is likely to be carried forward as a species of suspense account which must be liquidated not very long after peace. Moreover, international credit is likely to re-act on the situation in a prejudicial manner. Even already the financial system of Germany is more strained than appears on the surface. The fact is advantageous to us as belligerents, but it will probably be prejudicial to us not long after the re-establishment of peace. At present much of the inconvertible paper circulating on the continent does not affect us here. When the inflation has to be squeezed out after the war, a disturbance of credit is not unlikely.

"A special aspect of the problems under discussion is the provision of capital for re-starting of trades contracted by the war, and for the restoration of Belgium and other regions desolated during the progress of hos-

tilities. Chalmers, writing a hundred years ago, supposed that in cases of this kind 'in a very few years the recovery both of population and labor would be completed.'

"The explanation which he gave was far from satisfactory even for the time at which it was written, and it is still more deficient as applied to the present circumstances, when in industrial countries, fixed capital is much more important than in Chalmers' day. In the last quarter of a century, any great catastrophe, such, for instance, as the partial destruction of San Francisco by earthquake and fire, has been repaired with comparative ease by bringing capital from outside. But the waste of war renders capital exceedingly scarce; in fact, a famine of capital after the war has been predicted. Such an anticipation is over-pessimistic, but capital is likely to be obtainable for a time only with some difficulty. It is to be feared that after the war Europe will experience very considerable straits for several years to come. Not only must the waste of war be made good, but its evil legacy in inflated funded and floating debts must be gradually dealt with, lessening by reason of increased taxation the normal margin for new savings. Increased work and greater economy are the only remedies, aided by improved methods of production.

"It is to be hoped that some of the inevitable loss will be repaired in time by better methods of organization and by an accelerated rate of invention. The waging of a just war results in quickening of the national spirit. It forces a nation out of the easy and well worn paths of custom and convention. Thus out of all the suffering and all the loss, some good will come. The large proportion of our young manhood, which has gone to serve the country on the seas or in the field, and which returns having looked death in the face without being afraid, will not take up life where it was left. The noble qualities that have been evoked by the stress of battle will remain and will influence civil life during the next generation. The outlook will be both broader and also more simple."

Your correspondent has quoted somewhat extensively but not wholly prematurely from two important pronouncements. It must, of course, be recognized that the end is not yet, much water must first flow under London Bridge, many corpses choke the ditches in Flanders, and much blood trickle down to the waves of the Aegean before we can ever think the end is in sight. But just as the longest and weariest route march is cheered by the thought of rest and refreshment at *billets*, so in the months or years of a war, seemingly long drawn, it will do no harm to consider what fresh arrangements will be made when the time for that change of occupation which amounts to rest comes, so long as that consideration is of the serious order and takes full account of new difficulties to be faced.

The National Physical Laboratory

PROGRAM OF RESEARCH WORK FOR 1915-16

The following statement of work proposed for the current year in the Department of Metallurgy and Metallurgical Chemistry has been adopted by the General Board.

1. METALLURGY

Alloys Research.—The work outlined in the report for 1913-14 will be continued. The forging and rolling of a considerable number of light alloys will constitute a large part of the work for the year. The theoretical study of the aluminium-zinc-copper system, and the investigation into the permanence of light alloys will also be continued.

The work on the Effects of Stress on Steel at High Temperatures, on the Intercrystalline Cohesion of Metals, and on the Melting Point of Iron in Vacuo, will be continued with the new apparatus which has been obtained and set up for that purpose.

Acetylene Welding.—An investigation of acetylene welds will be carried out in conjunction with the Engineering Department.

Magnet Steels.—The research on magnet steels will be continued in co-operation with the Physics Department.

Volume Changes in Metals and Alloys on Fusing.—This research, which has been delayed owing to serious experimental obstacles, will be resumed, when circumstances permit, by means of new methods which have recently been devised.

Systematic Study of Steels.—The developments in the apparatus for taking thermal curves and for carrying out quenching and annealing which have taken place during the past few years, now render it eminently desirable that these new and more accurate methods should be utilized for a re-investigation of the thermal curves and micro-structures both of the fundamental carbon steels and of the more important special alloy steels derived from them. The latter study is particularly important in view of the increasing rôle which these special steels play in the investigatory test work of the department, particularly in connection with aeronautics. The systematic study indicated is, however, a large matter, involving a very considerable amount of work, and it is, moreover, correlated with the progress of two other researches, viz.: that on the melting point of iron in vacuo, and that on the effects of stress on steel at high temperatures. The present staff, and to some extent the equipment, of the department, do not permit of such an extensive investigation being taken up vigorously or with any hope of early completion. It is, however, so important that it is hoped to make a beginning as soon as circumstances permit.

II. METALLURGICAL CHEMISTRY

The study of analytical methods, and the further improvement of the methods in use in the Department, will be continued. Questions arising out of these matters will be investigated.

Investigations in conjunction with various industrial developments and problems arising out of war conditions will be undertaken.

With regard to electrotechnics and photometry, in regard to the latter the development of suitable substandards for use in the testing of gas-filled electric lamps is becoming an urgent matter. Experimental standards have already been made up for the laboratory by the Osram Lamp Works, and are being thoroughly tested before duplication into complete sets of standards. The cascade method adopted at the laboratory for the realization of candle-power in white light will be eminently suited for extension to these sources of light of still higher efficiency. Four sets of gas-filled standards are required to bridge the color interval between the present high temperature standards operating at 1.5 watts per candle and those which are required in the testing of nitrogen-filled lamps, and their preparation and standardization will be pushed forward during the year.

The experiments on the coefficient of reflection of the white cards used in photometry will be continued and extended to some other materials, a knowledge of whose characteristics in this respect will be of practical value.

If opportunity permits, it is hoped to repeat to a greater accuracy and to extend the scope of the experiments on the method of color identity for the estimation of the temperature of glowing bodies.

Measurements are being made at the instigation of the British National Committee of Illumination to determine the correct flame height of the Hefner lamp to give one British candle.

In regard to electrotechnics, some investigations on micanite are in hand and nearing completion, while the main research work of the year will be a continuation of the research work on buried cables.

Market Prices

SEPTEMBER, 1915

Copper opened £67.17.6 and was 68.5.0 on the 2nd, it then fell to £66 on the 6th, but recovered and continued to increase to 70.7.6 on the 21st and after a drop to 69.12.6 (23rd) rose to £71 on the 28th and closes £72.

Tin opened £151.10 and went better to £154.10 on the 3rd, then steady at 152.10.0 till the 10th, when it improved slightly to £154.10 on the 13th; since slightly lower, recovering to £154 on the 21st, then dropping to £150 on the 23rd, and subsequently recovering to £153.5 on the 28th, and closes weaker, £152.

Iron, Haematite opened 95/— and remained at this price throughout the month.

Iron, Scotch opened 70/9, and, after a rise to 70/10½ on the 3rd became easier till the 9th when 70/10½ again was reached. It was lowered on 15th and 16th to 70/—, but recovered to 70/9 on the 17th, dropping then to 70/3 on the 21st, and 70/— on the 27th; settlement, 70/9.

Cleveland opened 64/9 and went to 64/10½ on the 3rd, then falling to 64/4½ (7th) and recovering to 64/10½ on the 10th. On the 13th it was again 64/9 and 65/— on the 13th. It subsequently went lower to 64/3 on the 21st and 64/— by the 24th; settlement, 64/9.

Lead opened £23.5.0 and reached £23.15.0 by the 3rd and £24 by the 10th. It remained at this till the 21st when appreciated to £24.5 and was £25 on the 24th and closes £25.5.0.

Aluminium, 98-99 per cent.	£190	0	0
Antimony, black sulphide powder, cwt.	3	15	0
Borax, British refined crystal, ton	23	0	0
Copper sulphate, ton	27	0	0
Caustic soda, 70 per cent, ton	10	0	0
Ebonite rod, lb.	..	3	0
India rubber, Para fine, lb.	..	2	4½
Mica, in original cases, medium	3. 6 to	6	0
Quicksilver (Spanish), bottle	16	7	6
Sal-Ammoniac, ton	53	0	0
Sulphur, recovered, ton	8	0	0
Shellac lb.	..	..	5
Platinum, oz., nominal	9	10	0

Recent Chemical and Metallurgical Patents

Gold and Silver

Continuous Agitation.—Apparatus for continuous agitation of ore slime, as in the cyanide process, is shown in Fig. 1, being the patented invention of CYRUS ROBINSON of Mount Vernon, N. Y. It consists of a

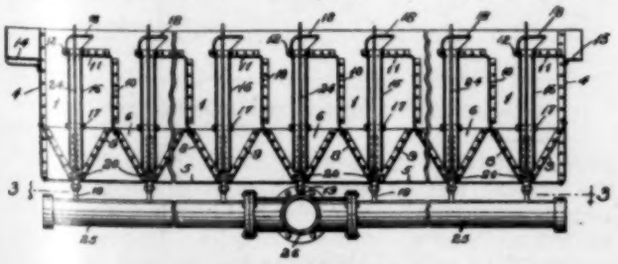


FIG. 1—AGITATOR

series of chambers connected, as shown. Extending beneath the chambers is a compressed air manifold with connections 19 to the bottom of each agitator. The latter has a central air-lift that is hooded in such a way that the discharged pulp is directed across an apron into the next agitator, and so on through the system, the pulp entering at 14 and issuing at 15. Regulation of the air used for agitation is accomplished by means of a nozzle 20 which has a beveled valve seat on which a conoidal valve closure rests. The closure has a depending finger that serves to protect it against lateral displacement and a vertical stem 24 which acts as a weight to hold the valve closed. (1,156,372, Oct. 12, 1915.)

Process of Cyaniding.—The diagram in Fig. 2 shows in considerable detail a patented procedure in cyaniding, by ROBERT S. TOWNE of New York City, and CYRUS ROBINSON of Mount Vernon, N. Y. The scheme

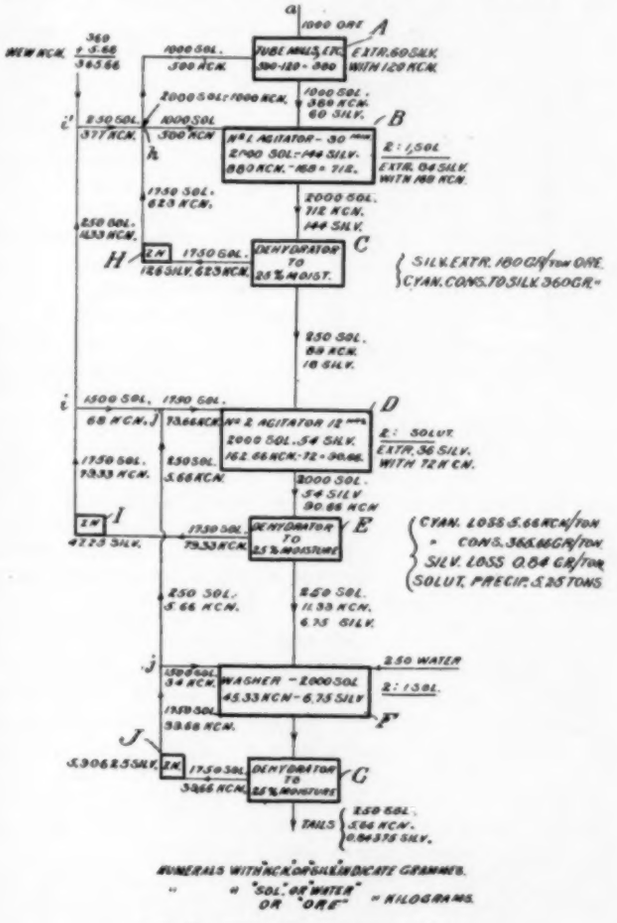


FIG. 2—PROCESS OF CYANIDING

is a multiple cycle process, the several cycles being successive in order and to a certain extent independent of each other. An object of the method is to effect extraction rapidly in a continuous process. It involves the use of a primary, relatively violent agitator and relatively strong solution, in combination with a secondary, relatively slow agitator and weak solution. The pulp is filtered between each step of the cycle so that the only solution flowing from one cycle to another is the amount carried with the dewatered ore. The quantity and strength of solution at various points can be regulated in accordance with the quantity and value of the ore, and the entire system maintained in equilibrium by the addition at an appropriate point of the proper quantity of fresh liquid and solvent. The diagram in Fig. 2 refers to the treatment of 1000 kilograms of ore containing 180 grams of silver per ton, for the extraction of which 360 grams of cyanide is required. (1,156,382, Oct. 12, 1915.)

Precipitating Metals from Solution.—A method of precipitating metals from solution, as, for example, gold and silver from cyanide solution by means of zinc dust, is patented by ROBERT S. TOWNE of New York

City, and CYRUS ROBINSON of Mount Vernon, N. Y. Previous methods of precipitating with zinc dust have made use of filter presses for the collection of the precipitant, but in this instance the inventors provide a settling or V-shaped receptacle, as shown in Fig. 3. The pregnant solution is mixed with zinc dust in any appropriate way and the mixture conducted through a pipe of considerable length before it reaches the separator, so that ample time for precipitation is allowed. The mixture is finally discharged from the pipe near the bottom of the receptacle 5, where the precipitate settles and is removed from the bottom. The solution freed from its precipitate flows upward and out of the tank by way of the launder 8. In order to prevent the upward flow and possible discharge of precipitate with the solution, a screen or grating 9 is placed across the tank, and on this screen is placed a layer of zinc in relatively finely divided form. This screen not only prevents the loss of precipitate, but also serves to obviate loss by re-dissolution of the precipitate while in the tank. (1,156,383, Oct. 12, 1915.)

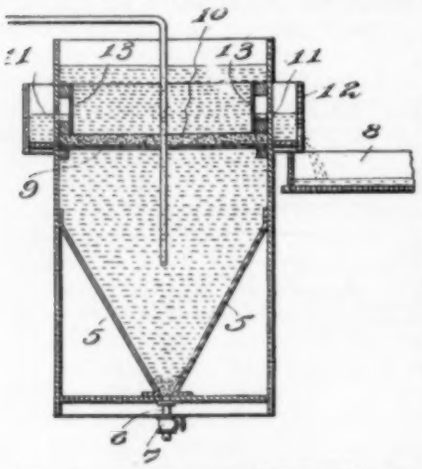


FIG. 3—DEVICE FOR PRECIPITATING AND SEPARATING METALS

Copper and Nickel

A process of separation of iron and nickel from copper in certain ores and metallurgical products is patented by WILHELM BORCHERS and EDUARD THILGES of Aachen, Germany. Experiments made by Lehmer at Aachen in 1906 showed that very pure metals could be produced by smelting sulphide ores with carbon and lime or limestone. According to this it would be easy to produce copper-nickel alloys by enriching a copper-nickel matte to a sufficient degree. But the market for copper-nickel alloy is not as good as for ferro-nickel, and since most copper-nickel ores and mattes contain iron sulphide in considerable quantity, it would be desirable to treat them for the production of ferro-nickel rather than copper-nickel. A high percentage of copper in the ferro-nickel, however, would be a disadvantage. The inventors have, therefore, discovered that copper can be separated from the greater part of the nickel and iron by melting copper-nickel matte with larger quantities of lime and smaller quantities of carbon than those used by Lehmer. In order to prevent the copper from separating in metallic form the matte is melted on a carbon-lined hearth of an electric furnace. Then an excess of lime is added, whereupon the iron and nickel sulphides are reduced and calcium sulphide is formed. The latter combines chiefly with the copper sulphide and prevents its decomposition. In this way it is possible by smelting comparatively poor copper-nickel mattes, to produce ferro-nickel, which is poor in copper and contains 15 to 20 per cent of nickel. (1,152,699, Sept. 7, 1915.)

Copper Hydrometallurgy.—The precipitation of copper from sulphate or chloride solutions by means of hydrogen sulphide and the reduction of cupric to cuprous sulphide with simultaneous generation of hydrogen sulphide for further use, form the basis of patents recently granted to RAYMOND F. BACON of Pittsburgh, Pa. The preliminary treatment of the ore to put copper in solution can be carried out in any well-known manner. The resulting solution is treated with hydrogen sulphide, with the precipitation of cupric sulphide and regeneration of the solvent. To the precipitate is added elemental sulphur in quantity equal to one-half the sulphur in the sulphide, and the mixture

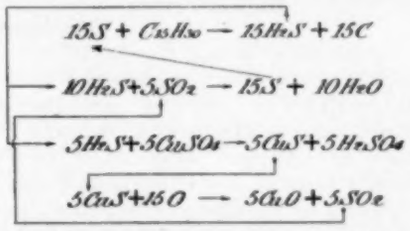


FIG. 4—COPPER HYDROMETALLURGY

is treated with a hydrocarbon body in a still at a temperature of about 300 to 360 deg. C., whereupon hydrogen sulphide is generated as a result of the reaction. Cuprous sulphide is formed and can be reduced by treatment in a converter or in any desired manner.

A related invention by the same patentee involves, first, the production of hydrogen sulphide by reaction between sulphur and a hydrocarbon; second, the use of a portion of the hydrogen sulphide to precipitate copper from a solution, and a portion to react with sulphur dioxide from roasting operations for the purpose of forming sulphur for utilization in the first step; third, roasting the cupric sulphide to oxide and treating the resulting sulphur dioxide as in the second step. The final products are carbon from the hydrocarbon treatment of sulphur and copper oxide from the roasting of cupric sulphide. These two products can be combined for the production of metallic copper. The relation of the different steps of the scheme are shown in Fig. 4. (1,151,234-5-6, Aug. 24, 1915.)

Electric Furnaces

Tapping Device for Electric Furnaces.—A simple method of tapping is patented by JAMES G. MARSHALL of Niagara Falls (assigned to the Union Carbide Company). When the tapping spout is made of metal and exposed to the molten material it usually melts out.

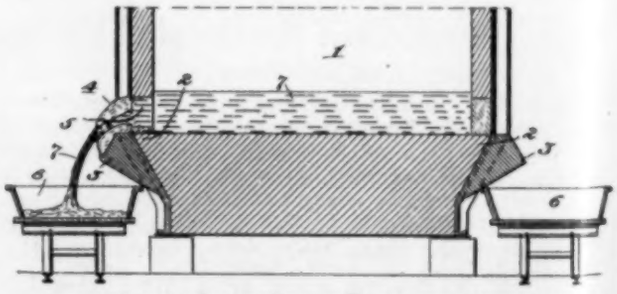


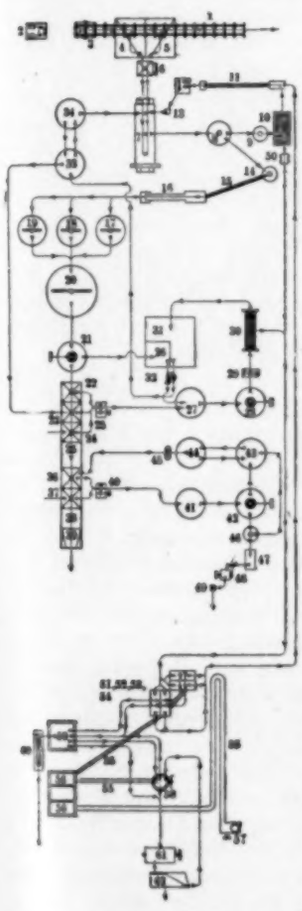
FIG. 5—TAPPING DEVICE FOR CARBIDE FURNACE

Repairs and renewals are inconvenient and costly. In the present method which is especially applicable to carbide manufacture, the furnace is made with an overhung hearth as shown at 2, Fig. 5. This allows the receptacle or chill 6, for the molten product to project under the furnace, and permits the tapping spout 3, to be merely a projection, so placed that it is not exposed to the direct heat of the molten material, but is protected by a layer of chilled material 4. In a carbide furnace the carbide chills rapidly on leaving the furnace, and an accumulation occurs at 4, thus preserving the spout. (1,149,203, Aug. 10, 1915.)

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt.—A paper by E. B. THORNHILL, metallurgical engineer for the Buffalo Mines at Cobalt, Canada, was presented at the San Francisco meeting of the American Institute of Mining Engineers, describing the process of recovering mercury from residues left



1. Incline track from L. G. Mill.
2. Compressed-air hoist.
3. Car elevator
4. Ore bin, jigs concentrate.
5. Ore bin, tables concentrate.
6. Charging hopper, cap'y 1,800 lb.
7. Tube mill, 5½ by 22, 27 rev. per min.
8. Settler, 8 ft. diam., 18 rev. per min.
9. Clean-up pan, 4 ft. diam., 17 rev. per min.
10. Amalgam bags and drain rack.
11. Mercury elevator.
12. Mercury reservoir.
13. Mercury meter box.
14. Auxiliary settler
15. Riffled launder.
16. Pulp elev. 60 ft., speed 313 ft. per min.
17. 18, 19. Dorr agitators, 10 by 10 ft.
20. Dorr agitator, 12 by 20 ft., collector.
21. Stock tank for filter, 10 by 10 ft.
22. Pulp tank, Moore filter.
24. Wash tank, water
25. Wash tank, water.
25. Vacuum pump, KCN, 6½ by 6 in.
26. Storage sump, pregnant KCN sol.
27. Storage tank, pregnant KCN sol.
28. Precipitation tank, KCN sol.
29. Triplex pump, 5 by 5 in.
30. Precipitate press, Perrin.
31. Storage sump, barren KCN sol.
32. Centrifugal pump for KCN sol.
33, 34. Storage tanks, barren KCN sol.
35. Working platform for filter.
36. Filter tank for mercury solv.
37. Filter tank for water wash.
38. Filter tank for acid wash.
39. Discharge chute for residues.
40. Vacuum pump, Na_2S, 6½ by 6 in.
41. Storage tank, pregnant Hg sol.
42. Precipitate tank, pregnant Hg sol.
43, 44. Storage tanks, barren Na_2S sol.
45. Triplex pump for barren Na_2S sol.
46. Wash tank for mercury precipitate
47. Drying pan for mercury precipitate.
48. Tube mill for mercury precipitate.
49. Bag for straining mercury.
50. Amalgam car.
51, 52, 53, 54. Retort furnaces.
55. Steel flues.
56. Dust chambers, 3 compartment.
57. Fan and stack.
58. Tilting furnace.
59. Refining furnace, reverberatory
60. Bullion molds and truck.
61. Ball mill at L. G. plant.
62. Wilfley table at L. G. plant.

FIG. 1—FLOW SHEET OF THE HIGH-GRADE AND MERCURY EXTRACTION PLANT

after amalgamating high-grade silver ore. A flow-sheet of the high-grade mill is shown in Fig. 1 and a chart of an amalgamation-barrel charge in Fig. 2.

In the amalgamation of high-grade ore in the Cobalt district considerable mercury is left in a residue, the metal being largely in the form of mercuric sulphide.

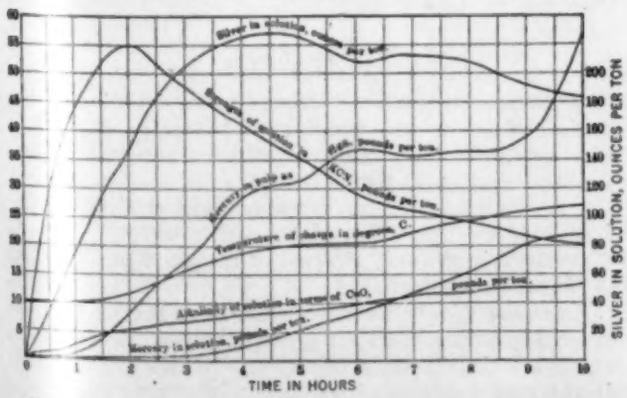

Silver in solution, ounces per ton
Strength of solution in KCN, pounds per ton
Mercury in pulp as KCN, pounds per ton
Temperature of charge in degrees, C.
Alkalinity of solution in terms of CaO, pounds per ton
Mercury in solution, pounds per ton
SILVER IN SOLUTION, OUNCES PER TON
TIME IN HOURS

FIG. 2—CHART OF AMALGAMATION BARREL CHARGE

The process of mercury recovery consists in leaching the mercuric sulphide from the residue by means of a caustic alkaline sulphide solution, and then precipitating mercury with aluminium.

The residue is caked on a Moore filter and washed free from silver solution with water. The filter is then lowered into a vat containing caustic alkaline sulphide, and this solution is drawn through the cake until all mercuric sulphide is dissolved. Usually 1 ton of solution is used per ton of residue. The resultant filtrate is agitated with aluminium, allowed to settle, and the clear solution decanted. The precipitate is dried, and the fluid mercury separated from the powdered portion, the latter being retorted.

The strength of solution is 4 per cent sodium sulphide and 1 per cent sodium hydroxide. The precipitant is a waste product from aluminium casting foundries, and about ¼ lb. is used per pound of mercury precipitated. Caustic soda is regenerated in the reaction, so that the consumption is but 1/10 lb. per pound of mercury recovered. Sodium sulphide also is regenerated, but there is a mechanical loss of about 20 per cent. The cost of recovering mercury by this process has been about 13 cents per pound.

Effect of Carbon or Graphite on Auriferous Cyanide Solutions.—In the June issue of this journal we presented an abstract of a paper by W. R. Feldtmann on the precipitating action of carbon in contact with auriferous cyanide solutions, originally presented before the Institution of Mining and Metallurgy. In *Bulletin* No. 130 of the same organization we note the following discussion of the subject by PAUL T. BRÜHL, who gives his experience with graphitic ore at the Prestea Block A mine, Gold Coast, Africa. The graphite had the following composition:

Carbon (with S and As)	11.06%
Silicious residue	75.00%
Fe_2O_3	6.56%
Al_2O_3	1.20%
CaO	4.50%
MgO	1.74%

The cyanide solution tested was the same in all cases, and assayed:

Potassium cyanide	0.20 %
Protective alkalinity	0.026%
Gold	61s. 4d.

Experiment 1—Two hundred and fifty grams of the mine graphite, which assayed 51s. 4d., and contained a little iron pyrites and mispickel, were agitated for four hours with 500 c.c. of the cyanide solution. The solution after filtering gave the following analysis:

KCN	0.07 %
P.A.	0.004%
Gold	0s. 4d.

Experiment 2—Two hundred and fifty grams of mine graphite, 50 grm. low grade concentrates, assaying 47s. 6d. per ton, and 600 c.c. of the solution were agitated together for four hours. The solution after filtering gave the following analysis:

KCN	0.07 %
P.A.	0.005%
Gold	1s. 6d.

Experiment 3—Two hundred and fifty grams of mine graphite, 50 grm. rich concentrates, assaying 50s. per ton, were agitated together for four hours. The solution after filtering gave the following analysis:

KCN	0.03 %
P.A.	0.003%
Gold	11s. 6d.

"These experiments indicate the powerful precipitating effect of the mine graphite. The solutions were progressively richer in gold because the sulphides were progressively richer, and the KCN had dissolved to some

extent the gold contents of the sulphides. Solution and precipitation of the gold take place simultaneously, the latter action being the more pronounced of the two in the early stages of the agitation.

"The fact, noticed by Mr. Feldtmann, that there is a load point at from eighteen to twenty-four hours beyond which precipitation is resumed, is one which I also noted when experimenting with slimes. In these experiments the most economic extraction was obtained from each particular charge by agitation for a certain definite number of hours, usually about eighteen; the time necessary varied and depended on the charge. Further agitation caused higher residue values."

In *Bulletin* No. 129, I. M. M., F. WARTENWEILER confirms Mr. Feldtmann's conclusions that the precipitated gold is not soluble in any of the strong oxidizing agents, such as chlorine or bromine, and that these do not inhibit precipitation by graphite. Search for other solvents showed that neither sodium hyposulphite nor sodium thiocyanate were effective in redissolving precipitated gold or in preventing precipitation. Mr. Wartenweiler found, however, as did Mr. Feldtmann, that sodium sulphide (0.30 per cent Na_2S) did dissolve precipitated gold in post treatment of residues from cyanidation, and that copper in the form of shavings gave promise of affording a suitable solution precipitant. He found further that an extra treatment could be avoided by adding the alkaline sulphide directly to the cyanide solution for original treatment, and get the same results as by cyanidation plus subsequent treatment with alkaline sulphide. Zinc forms a good precipitant for the solution provided no free Na_2S is left in solution. A peculiar feature of the use of the sodic-sulpho-cyanide solution is that redissolved gold remains in solution for many hours while in contact with the schist, although sulphide may no longer be indicated by qualitative tests.

Comparing the relative gold solvent power of Na_2S solution and the sodic-sulpho-cyanide solution the following empirical statements may be offered:

(*a*) With a clean carbonaceous schist carrying considerable precipitated gold the simple Na_2S solution is the best solvent.

(*b*) With a works concentrator tailing carrying the fairly constant quantity of gold precipitated by the contained schist, the two solutions dissolve equally well the precipitated gold which varies from 2s. to 4s. per ton.

Lead and Zinc

Selective Flotation of Sulphide Minerals.—An Australian invention, recorded in the *Australian Statesman and Mining Standard* for July 22, 1915, provides for the separation of zinc sulphide from the sulphides of lead and iron in the flotation process, by adding to the pulp certain mineral salts which can be decomposed by the acid used in the process, liberating a reducing gas. Thus the inventor, Leslie Bradford of Broken Hill, Australia, uses thiosulphates, sulphites and bisulphites, which, in reaction with sulphuric acid, liberate sulphur dioxide. Or, the gas may be introduced into the pulp in any other manner with equally good results. The net effect is to produce a wetting of the zinc sulphide, causing it to remain in the pulp and be discharged with the gangue, while the lead and iron sulphides float. The separation of the blende from the gangue is then accomplished in a separate operation.

Crushing and Grinding

Scoop Discharges for Tube-Mills.—Though the idea of scoop discharges for tube-mills has received a great deal of attention recently, it is not wholly new. In a paper on the subject by WALTER R. DOWLING, published in the March, 1915, *Journal* of the Chem., Met. & Min. Soc. of S. Africa, the author illustrates "a scoop discharge with radial arms and perforated plate of 2-ft. diameter," originally installed at the Knights Deep. The drawing bears the date of April 20, 1907. This device was used on account of trouble experienced with backflow of pulp at the feed end of the tube, and was designed to lower the internal level of the pulp. Scoop discharges of improved design have been used at this plant ever since their first introduction.

Some time later Dr. W. A. Caldecott made experiments with an apparatus in which the scoop arms were long enough to cover an effective diameter of 50 in. The scoop consisted of four arms attached to a plate, which, when bolted to the discharge end of a tube, provided four chambers that brought up their quota of pulp as the mill revolved and delivered it to the outlet. A perforated plate was bolted in front of the scoop to prevent the discharge of pebbles. For purposes of experiment in determining the most effective depth of pulp in the mill, without varying the diameter of scoop arms, different effective diameters of perforated plate were used by covering annular areas of the outer portion of the plate. Tests were made at different times at different mills, in order to arrive at average conditions from which conclusions could be drawn.

Assuming that the present average duty, under ordinary conditions, of a standard 5½-ft. by 22-ft. tube-mill is 120 tons of minus 90-mesh product per twenty-four hours, the performances recorded in the tests with scoop discharge showed considerable improvement, the comparative duty being 143 tons. Increase of pebble load to a height materially above the axis of the mill is of little assistance, since the power consumption and pebble wear increase out of all proportion to the increase in quantity of minus 90-mesh product. Thus, in one case, a pebble load of 16.08 tons took 20.04 hp. more to produce only 3.22 tons more than a pebble load of 12.84 tons. The most effective diameter of scoop was found to be 24 in., considering the results obtained and the wear of liner and pebbles. With larger diameter scoop the wear is excessive. There does not appear to be any saving in power in the use of scoop discharge; but the real advantage lies in increased capacity, which is such that five tubes with scoop discharge will do the work of six trunnion discharge mills. Small sand feed decreases the tons of minus 90-mesh product; and as increased sand feeds up to 500 tons do not increase the power consumption of the mill, there is no reason why big feeds should not be given. The most suitable feed of sand per twenty-four hours as a pulp containing 40 per cent moisture, for a 5½-ft. by 22-ft. tube-mill appears to be about 400 tons. The reason for the increased capacity afforded by the scoop discharge is that the level of pulp is lowered and pebbles strike against each other, crushing the adhering grains of sand, rather than falling into a mud bath as under ordinary conditions.

Gas Producers

Wood-Gas Generators.—In the reconstruction of the surface plant at the Sons of Gwalia mine, Australia, wood-gas generators were substituted for charcoal-gas producers and have proved satisfactory and economical. The producers are of the Commonwealth modified downdraft type. Operating data are given by Mr. A. WAUCHOPE in the July *Journal* of the Chamber of Mines of Western Australia.

Firewood in 5-ft. lengths is fed in at the top and the charcoal produced forms a glowing porous mass through which are drawn the products of distillation (i. e., hydrocarbons, steam, etc.) together with the necessary air, thus forming a gas consisting of carbon dioxide, carbon

monoxide, hydrogen, methane and nitrogen. A small amount of oxygen is usually also present in the gas. The following is an analysis of the composition of the gas under normal conditions:

Carbon dioxide (CO_2)	9.7 per cent
Carbon monoxide (CO)	19.5 per cent
Hydrogen (H)	14.1 per cent
Methane (CH_4)	2.1 per cent
Oxygen (O)	0.1 per cent
Nitrogen (N), by difference	54.5 per cent

Calorific value, estimated on composition, 125 B.t.u. per cubic foot.

The charcoal rests on an ordinary fire-bar grate through which the ashes and gas pass into a water-sealed hopper bottom. On a level with the grate there are several doors in the sides of the producer through which the grate is raked free of clinker and ashes. The opening of these doors does not appreciably affect the value of the gas within reasonable limits of time. The volume of gas produced is regulated by the suction of the engine which exerts a pull below the fire grate of the producer ranging from 2 in. to 5 in. of water.

Power costs on those sections of the cyanide mill driven by gas on a horsepower basis are 66 per cent of the power costs of those sections driven by steam. Mulga firewood, having a calorific value of about 7000 B.t.u., is consumed at the rate of from 2 to 2.2 lb. per brake-horsepower-hour.

Fused Silica Ware Made in United States

Things are moving in the chemical industries of the United States. The latest example of the establishment of a new American industry is the announcement that the firm of Lenz & Naumann, Inc., of New York City, has acquired all the American and Canadian patents and manufacturing rights for the system of Dr. Voelker and is equipping now a plant for manufacturing "quartz glass."

Dr. A. Voelker of Cologne, Germany, has been working on the problems of making fused silica ware for many years, as in 1899 he perfected his process and since then, in years of untiring efforts, he slowly developed his large-scale manufacturing methods which to-day make it possible to produce not only ordinary laboratory ware, but articles in such large sizes and such shapes as were considered impossible in former years. The Voelker process permits of the manufacture from ordinary quartz sand not only of opaque but also of transparent (semi-translucent) and translucent quartz. Waterclear quartz glass is made from rock crystal.

This fused silica ware is being introduced by Messrs. Lenz & Naumann under the name of Silicium Dioxyde.

Chain Screen Doors for Oven and Furnace Openings

Everyone who has had to work around ovens and furnaces knows what discomfort has to be endured by the attendants from the heat and gases coming out through the uncovered opening, when the doors are thrown open, and, further, what a chilling effect the inrushing cold air has upon the interior of the furnace and its contents. Doors, however, must be opened; be it to renew the charge, manipulate the contents, or study the interior conditions.

The Wiegand chain screen doors, in the form used mostly around glass, metal and chemical furnaces, consist of a multitude of freely hanging individual strands of steel chain suspended close together from a steel bar in a manner to form a continuous sheet or curtain of chain, not unlike the familiar Japanese screen.

This curtain of chain, hung before the uncovered opening to a furnace and looking like a coat of mail,

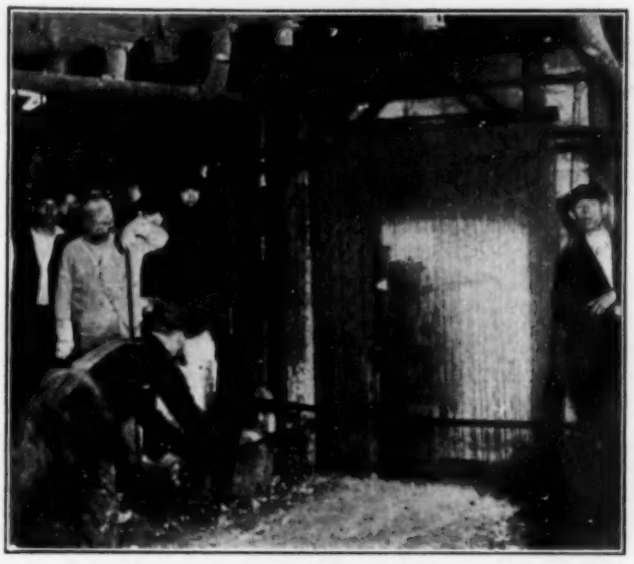


FIG. 1—WORKMEN PROTECTED BY CHAIN SCREEN IN GLASS WORKS

effectively hinders the heat, glare and gases from leaving the furnace and the cold air from entering. The loosely hanging strands of light chain are parted with ease and pressed aside by the tools or other objects projected into the furnace, only to fall together again when entrance has been effected.

The holes in the links of chain permit an unhampered view of the interior—in fact, a better survey than under ordinary conditions may be obtained, as the glare is toned down and the effect is similar to looking into the furnace through a piece of wire gauze.

In some plants, when it is necessary to work in front of the uncovered oven or furnace, the workmen are obliged to stand back a great distance from their work and to protect their bodies from the heat and glare with large sheet iron shields which are supported by one hand, while the other manipulates the tools. This seriously handicaps the workman and cuts his efficiency in half. In those plants where the new chain screens have been employed the workman has both hands available for his work and may, with comfort, freely manipulate his tools while standing within a few inches of his work.

FIG. 2—REMOVING HOT POT FROM GLASS FURNACE THROUGH CHAIN SCREEN

When it is necessary to remove a damaged pot in a glass works the chain screen shield is lowered in front of the wall enclosing the pot that is to be removed. The wall is then torn down, and the hot blocks composing the same are dragged through the chain shield and set aside. Now the broken pot is clearly seen in the furnace behind the screen, which closes the opening to the furnace, and the workmen may operate without discomfort close to the pot itself. The absence of heat and glare, coupled with the ability to see clearly what they are doing, enables the workmen to quickly pry up the pot, shove the nose of the pot truck under the pot and drag it through the perfectly flexible and penetrable shield out of the furnace. The strands of chain offer no obstruction to the passage of the truck and pot, immediately falling together again when the pot has passed, closing the opening; thus keeping the heat within and the cold drafts out of the oven or furnace.

When the truck carrying the new pot glowing with heat arrives before the screen shield, protecting the furnace, it is pushed boldly through the same. When once the truck and its pot have passed through the chains the latter close together, protecting the workmen from the radiant heat and the new pot and the whole furnace from drafts of cold air.

The doors are supplied in "automatic" and "non-automatic" forms. The automatic is employed mostly on boiler furnaces, where the opening of the ordinary fire door causes the screen to unroll from a cylinder hung above the furnace opening. The non-automatic, which is much used on ovens and other furnaces, is much simpler in construction. The chains forming the screen are suspended from a bar and may be raised out of the way by a hand chain running through a pulley overhead. When needed the screen is lowered into supporting brackets fixed at the sides of the opening.

In the glass factory, where a very large screen is needed and not more than one opening is made in the furnace at one time, a single screen is made to do duty for a whole furnace. Wherever the wall is to be torn down for the removal of a pot the screen is hung up by its pulley.

These chain screen doors are the result of several years of experimentation, having been first adapted for use on boiler furnaces, where they have been in practical operation for three years. They are made by the E. J. Codd Company, 700 South Caroline Street, Baltimore, Md.

Bihn-Jones Automatic Air Device for Raising or Handling Large Quantities of Liquids

In chemical and industrial plants there are many processes in use requiring the raising or handling of large quantities of liquids, and the use of compressed air has long been considered in this connection as a useful and necessary means of power transmission.

In making use of this compressed-air power for handling liquids, it is necessary to have a blow-case or air-tight chamber into which the liquid is admitted (usually by gravity), and from which it is forced to its destination by the compressed air, the latter being also admitted to the blow-case at necessary intervals. The process consists, therefore, in admitting liquid until the blow-case is filled, then turning off the liquid and turning on an air valve which admits the air. This operation is repeated continuously as long as liquid is to be lifted.

It has long been the effort of inventors to obtain an automatic device which will turn on and off the valves necessary for this operation, and a number of apparatus have from time to time appeared which have been claimed to accomplish this purpose. The chief objection, however, to these apparatus has always been their complicated construction, allowing too great an opportunity for trouble in the mechanical operation.

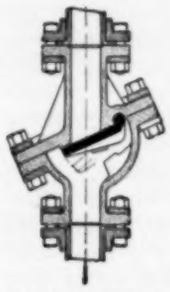


FIG. 1—ANY-POSITION CHECK VALVE

The latest of these devices, known as the Bihn-Jones automatic air device, seems to have overcome all these objections and is very simple in construction and operation. As it attracted considerable and well deserved attention at the recent National Exposition of Chemical Industries in New York City, a somewhat more detailed description should be of interest.

The apparatus consists of two parts. The first is an any-position check valve (Fig. 1), which is composed of two essential parts, namely, a body which contains the valve seat and a flapper which swings on trunnions, thus constituting a valve which will operate in any position. The second piece of apparatus is the automatic air device proper. It is composed, first, of a body in which is formed the seat of the valve, through which the ports for the admission of compressed air open, the upper portion of the body also containing the exhaust port. Secondly, a disk of a cone and piston type is provided which has two seats, at the bottom a cone

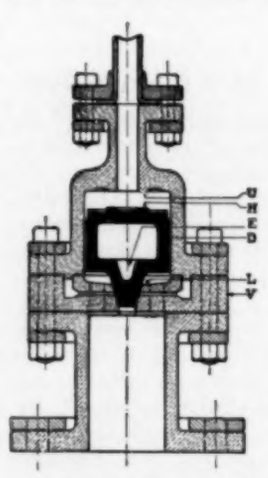


FIG. 2—POSITION OF AIR DEVICE WHEN TANK IS FILLING

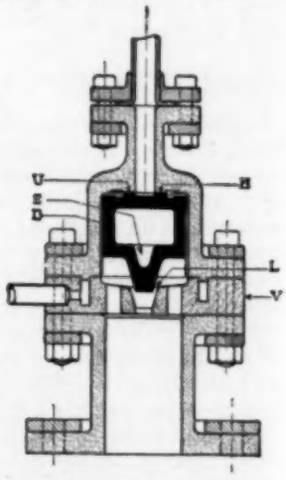


FIG. 3—POSITION OF AIR DEVICE WHEN TANK IS EMPTYING

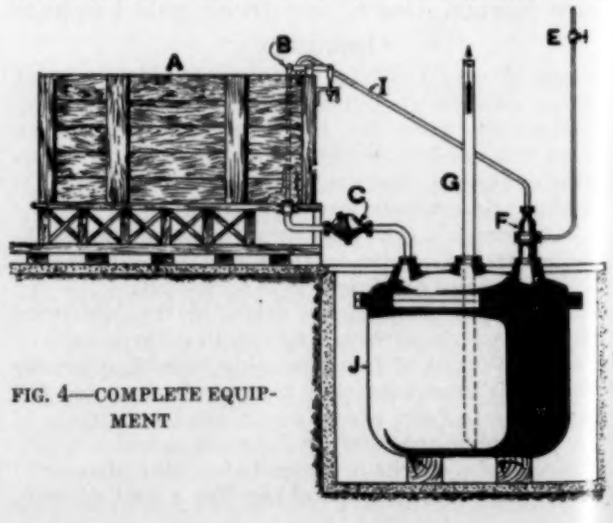


FIG. 4—COMPLETE EQUIPMENT

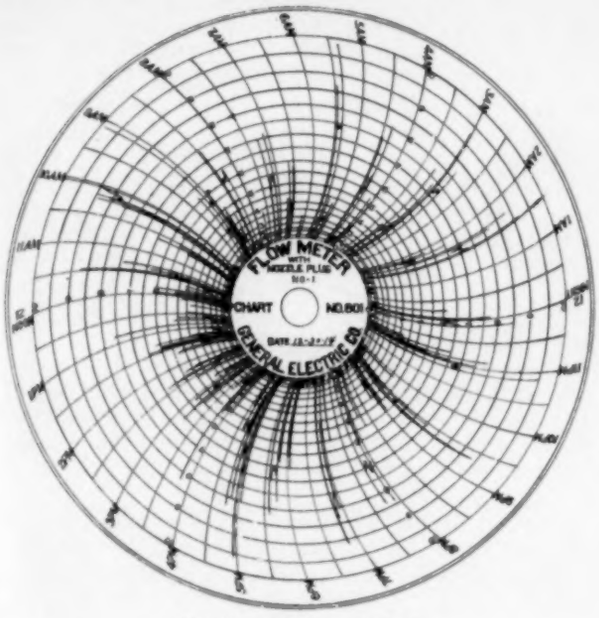


FIG. 5—RECORD CHART

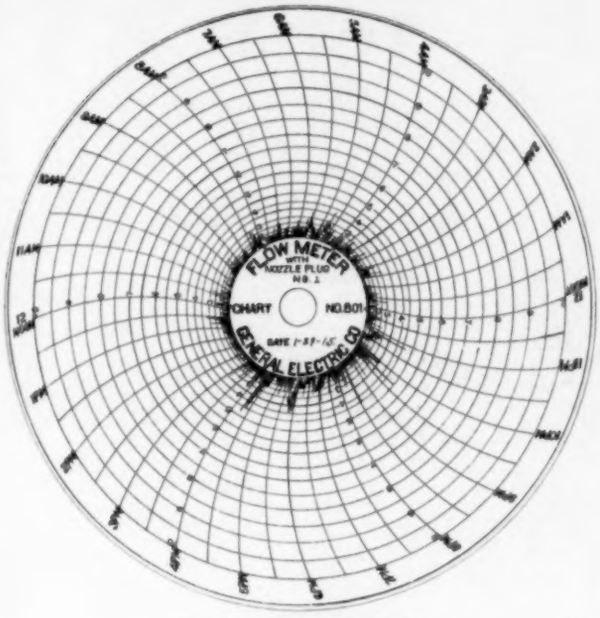


FIG. 6—RECORD CHART

which closes the compressed-air ports and at the top a flat seat which closes the exhaust port. These devices are illustrated in Figs. 2 and 3 respectively.

DETAILED ACTION OF DEVICE

Parts of Air Device.—The air device consists of the following parts:

1. Valve body *V*, which contains a seat *L* controlling the ports for the admission of compressed air, and a seat *U* for controlling the exhaust ports.

2. Disk *D*, of the cone and piston type, the cone *E* being at the base and completes the compressed-air valve. At the top of the disk *D* is a seat *H*, which completes the exhaust-air valve.

The device is actuated by the liquid from storage tank *A*, which flows through the check valve *C* into the blow-case *J*. This liquid rises until it reaches the cone of the disk *D* of the automatic-air device *F*, which it raises, thus permitting compressed air to enter.

The sudden release of the compressed air forces the disk *D* upward against its upper seat *U*, thus closing the exhaust port to vent pipe *I*.

The air pressure is thus prevented from escaping and the pressure exerted on top of the liquid closes the check valve *C* and forces the liquid through discharge pipe *G* to its destination.

When the level of the liquid reaches the opening of the discharge pipe at bottom of blow-case, the compressed air rushes out with the liquid, thus clearing the discharge pipe. The result of the air rushing out the discharge pipe causes a considerable drop of pressure within the blow-case, thus allowing the disk *D* of the air device to drop to its lower seat *L* due to its own weight, simultaneously cutting off the supply of compressed air and opening the exhaust port.

The check valve *C* in the supply line has remained closed during the emptying of the blow-case, but immediately upon the reduction of pressure within the blow-case the check valve opens (due to the head of the liquid in the supply line) and allows the blow-case to refill.

The free air or gas from liquid displaced from blow-case due to refilling is discharged to atmosphere through vent *I*.

The flow of liquid from tank to blow-case may be regulated by a plunger and thumb screw arrangement, as shown at *B* in Fig. 4, or by means of a stop valve. Accordingly, by regulating the supply, the contents of blow-case may be automatically discharged in any specified time.

It will be noticed from the above illustration that the apparatus is entirely outside of the blow-case, no part of it extending within the same, but resting upon a flange collar above the case. It can therefore be readily removed for examination, and its ease of operation may be readily understood.

Aside from being entirely automatic in operation and simple in construction, this device is claimed to reduce considerably the consumption of compressed air which is known to be quite expensive. The charts, Fig. 5 and Fig. 6, were made in a plant where forty blow-cases are in use, seven of which were equipped with the Bihn-Jones automatic air device at the time the test was made. The heavy irregular lines indicate the exact travel of the needle, which indicate the comparative consumption of compressed air. Fig. 5 was produced when the blow-cases were operated by hand, that is, compressed air turned on and off each time the blow-case was discharged, while Fig. 6 illustrates the consumption of air by the seven blow-cases which were equipped with the Bihn-Jones device.

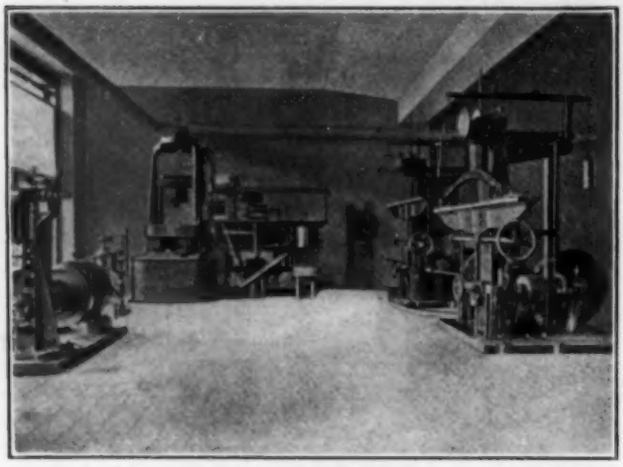
The saving in compressed air is further illustrated by the fact that in the above-mentioned case, in which the forty blow-cases, of 3 to 5½ tons capacity, are being filled and discharged continuously day and night, after the installation of the seven Bihn-Jones automatic air devices one of the compressors was shut down and has since been kept as a reserve unit.

This device is claimed to be particularly adaptable to conditions wherein large bulks of liquids are to be handled, and by properly selecting the material from which it is made it will handle with equal efficiency corrosive material, oily liquids, and, in fact, any material which may be readily caused to flow by compressed air. Its most extensive application is found in handling of sulphuric acid in fertilizer and acid manufacturing plants.

The device is being manufactured by The Schutte & Koerting Company of Philadelphia, Pa.

The Scientific Study of Steel

With a view to taking up the study of the numerous problems in iron and steel on a broader ad more complete scale the Western Spring & Axle Company, whose



TESTING MACHINES IN PHYSICAL LABORATORY

main offices are at Cincinnati, Ohio, have constructed new chemical, physical and microscopical laboratories at Detroit, Mich. This central group of laboratories is larger and more completely equipped than any single member company would be justified in maintaining.

The *chemical laboratory* is completely equipped for the analysis of each shipment of steel as it comes from the mill.

The *physical laboratory* for heat treatment and physical testing contains electric furnaces and a number of testing machines, as shown in the illustration. The machine on the extreme left is a special Upton-Lewis fatigue testing machine. This machine will take any size test bar up to 2 in. wide and ⅜ in. thick and subject the metal to any desired fiber stress and vibrate it at a maximum rate of 1000 r.p.m. It also automatically describes a curve showing the relation between fiber stress and the number of vibrations and the behavior of the steel during the test.

The *microscopical laboratory* is completely equipped for the photographing and examination of metal test specimens, including a recently perfected "inverted" microscope with camera attached having a magnifying range of 20 to 1000 diameters.

The laboratories were instituted not only for member companies, but in their careful planning and complete equipment there has been the thought of a larger usefulness, through co-operative effort with engineers in solving the many problems that arise in this field. The laboratories are in charge of Mr. Erwin C. Arndts, and the company extends a cordial invitation to all who are interested to visit the new laboratories at Detroit.

Personal

Mr. Charles W. Baker will take charge of the recently opened New York office of the American Zinc, Lead & Smelting Co. in the Equitable Building. Mr. Baker was recently elected a director of the company.

Mr. George B. Clark has sold his Pueblo, Col., assay office and has taken a position as assayer and chemist for the River Smelting and Refining Company, Florence, Col.

Mr. John Gross, mining and metallurgical engineer, has recently opened on office at 523 McPhee Building, Denver, Col.

Mr. E. M. Hamilton, Mr. F. A. Beauchamp and **Mr. S. E. Woodworth** have gone into business together as metallurgists. Their offices will be at 419 Embarcadero, San Francisco.

Mr. F. E. Marcy has given up the management of the Salt Lake City office of the Mine & Smelter Supply Company, and will devote his time to the promotion of the Marcy mill, which is made by that company.

Mr. F. E. Parks will become manager of the Minnequa works of the Colorado Fuel & Iron Company, succeeding J. B. McKennan, who becomes general manager of the company.

Mr. James Coles Roberts of the U. S. Bureau of Mines, Denver, Col., has been selected for the Joseph Austin Holmes Professorship of Safety and Efficiency Engineering at the Colorado School of Mines.

Mr. L. Selmi, chemist and metallurgist of the Otis Steel Co., Cleveland, Ohio, has resigned to take charge of the chemical and physical laboratories of Corrigan, McKinney & Co. of Cleveland, a new steel plant.

Mr. R. G. Williams, safety engineer of the Norton Company, Worcester, Mass., has succeeded Mr. F. C. Schwedtman as chairman of the committee on accident prevention of the National Association of Manufacturers.

Mr. F. O. Williamson, representative of the Hardinge Conical Mill Co., T. L. Smith Co., and Smith Engineering Works in Chicago, is taking a trip through the Western mining districts.

Book Reviews

Molecular Association. By **W. E. S. Turner.** Octavo, 170 pages, 6 illus. Price, $1.40. New York: Longmans, Green & Co.

This is one of the monographs on inorganic and physical chemistry, edited by Dr. Alexander Findley. The topics principally dealt with are molecular association in gases (a comparatively simple field), of dissolved substances (an uncertain subject), the molecular complexity of the liquid state (an almost unknown territory), and of the solid state (a collection of speculations). While the monograph may be styled "satisfactory" because it presents nearly all that can now be said on the subject, yet it is necessary to say that the subject itself (aside from the topic of gases) is in a highly unsatisfactory condition, and there is not much gratification in store for the reader who peruses the book with the thought of learning something definite about the molecular state of solutions, liquids and solids. The author himself appreciates this situation, saying at one place: "There is great need of some guiding and reconciling principle like Avogadro's hypothesis." The reader will agree with him.

Methods in Metallurgical Analysis. By **Charles H. White.** 356 pages, 106 illus. Price, $2.50. New York: D. Van Nostrand Company.

Professor White writes from the standpoint of his experience in the Harvard School of Mining and Metallurgy. This is a text-book of quantitative analysis for metallurgical students. It is plainly written, in simple, direct style, giving the information needed in the fewest possible words.

It is an admirable repository of methods, so abridged as to read almost like recipes. In fact, this is the chief lack of the book, the frequent absence of reasons for the steps described. Assuming that the book is to be used under the careful supervision of a capable instructor, who will explain the *why* and *wherefore* (and thus, in reality, instruct and educate the student), it is an admirable laboratory manual; but for a man working without such assistance it would be an unsatisfactory guide.

It deserves a place in the commercial laboratory analyzing ores, metals or alloys, but the information is rather elementary for this purpose and must be used discriminatingly.